A PRE-FEASIBILITY STUDY FOR THE PHOSPHATE DEVELOPMENT PROJECT

THE REPUBLIC OF ZAMBIA

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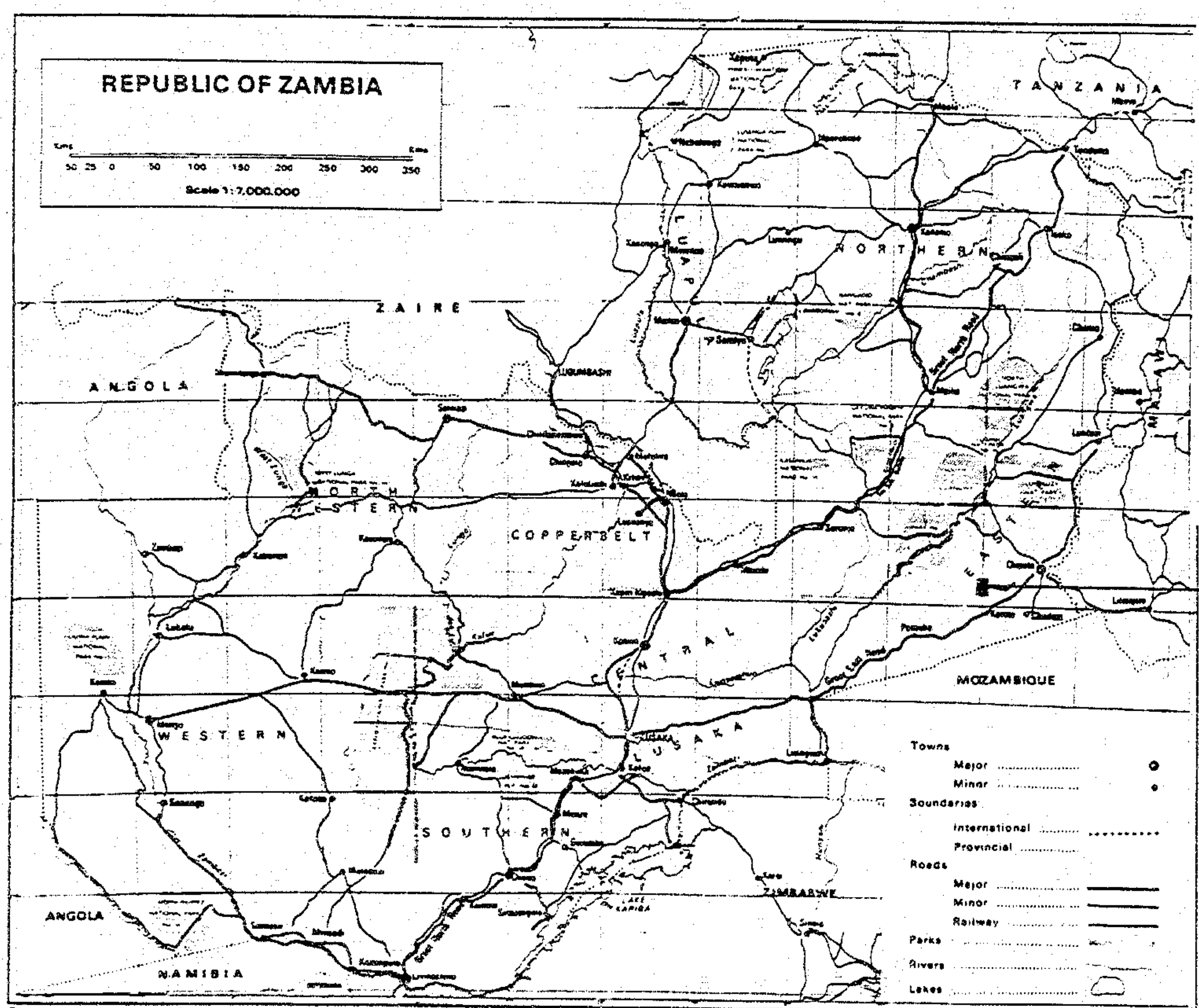
1985

JAPAN INTERNATIONAL COOPERATION AGENCY



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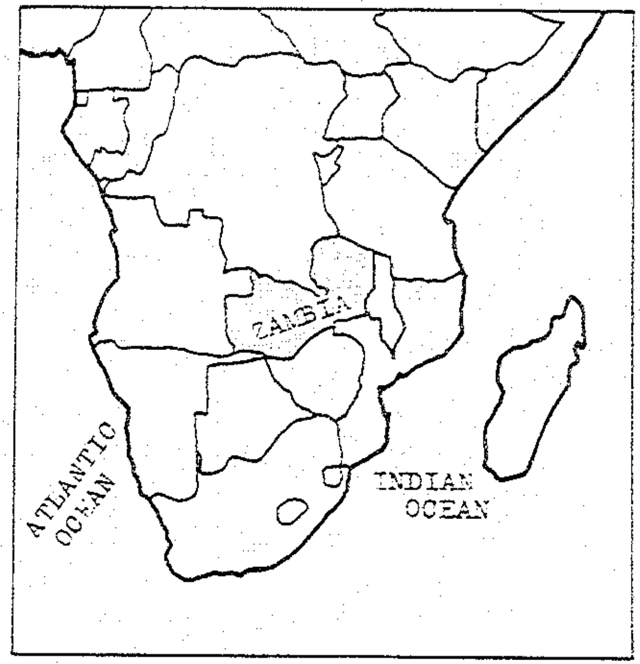

LOCATION MAP

PREFACE

It is with great pleasure that I present to the Government of the Republic of Zambia this report on Pre-feasibility Study of the Phosphate Development Project.

This report is based on the result of a field survey which was carried out from June to July–September, 1984 by a Japanese survey team commissioned by the Japan International Cooperation Agency (JICA), following the request of the Government of the Republic of Zambia to the Government of Japan.

The survey team, headed by Mr. Takashi ONO, had a series of discussions on the Project with the officials concerned of the Government of Zambia and conducted a wide-ranging field survey and data analyses.

After the team returned to Japan, further studies were made and the present report has been prepared.

I sincerely hope that this report will be useful as a basic reference for realization of the project and thereby contribute to the promotion of friendly relations between our two countries.

I wish to express my deep appreciation to the officials concerned of the Government of the Republic of Zambia for their close cooperation extended to the Japanese team.

June, 1985

Keisuke Arita

Keisuke ARITA
President
Japan International Cooperation Agency

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Abstract

Since 1980, a detailed prospecting programme has been carried out by MINEX, over an area of apatite occurrence called the Chilembwe Prospect on the land of Sinda West in the Eastern Province.

Pursuant to the request made by the Government of the Republic of Zambia, the Japan International Cooperation Agency decided to implement a pre-feasibility study for the phosphate development project.

After the field exploration work and the laboratory investigation, a mining plan was designed to produce annually a sum of 35,000 tonnes of 30% P_2O_5 concentrates over a period of fourteen years. With the estimation of the capital cost and the operation expenses, the internal rates of return were calculated in the financial and the economic evaluations. A marginal profit can be expected on a private mining enterprise and somewhat a higher contribution to the national economy is estimated.

Establishment of the apatite mining requires the existence of the phosphatic fertilizer plant in the country. The overall economic evaluation on the phosphatic industry should be made in connection with a feasibility study of the fertilizer plant.

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1. INTRODUCTION

1.1 Background of the Project

The Republic of Zambia is one of the leading copper and cobalt producing countries of the world and its mineral exportation is virtually the best means to obtain foreign currencies. However, the stagnant copper price prevailing for a long time has aggravated the balance of foreign currencies of the country. In view of this situation, the Republic of Zambia is now raising the banner for an increase in its agricultural productivity as one of its priority policies in its Third National Development Plan 1979 ~ 1983.

At present, the compound mixture of fertilizers which represent 60% of the national consumption is imported from abroad and only the straight nitrogenetic fertilizer is being produced to cover 25% of the domestic demand.

1.2 Phosphate Resources in Zambia

The Government of the Republic of Zambia is aiming at the home production of phosphatic fertilizers as one of its measures to increase its agricultural potential. MINEX, the mineral exploration department of the Zambia Industrial & Mining Corporation Limited, has been engaged in a search for phosphates and two types of igneous phosphates have been recognized. Syenite-related phosphates are found in Chilembwe in the Eastern Province and similar rocks are reported from the northwest of Mumbwa in the Central Province. Phosphates related to carbonatite are located at Kaluwe, 220 km east of Lusaka. Similar deposits are found at Nkombwa, but ore mineral of this place is said to be isokite, of which no commercial utilization is achieved. Besides these two types, sedimentary phosphates are also being investigated by MINEX.

1.3 Objectives of the Study

Highly anamalous value of phosphate was recorded during a reconnaissance survey in late 1970's, and MINEX has delineated several promising areas known as Chilembwe on the land of Sinda West in the Eastern Province. Pursuant to the request made by the Government of the Republic of Zambia, the Japan International Cooperation Agency dispatched a preparatory mission in October 1983, and decided to implement "the Pre-Feasibility Study for the Phosphate Development Project." The Project intends to study the feasibility of development of apatite deposits in Chilembwe as a step toward the realization of the home production of phosphatic fertilizers.

1.4 The Scope of Work

The scheme of the present investigation was divided into two stages.

The process of the first stage comprises

(1) confirmation of the ore reserves by additional drilling, and

(2) an beneficiation test of ore minerals.

In addition to these, preliminary investigation of magnesium resources was recommended for the provision of raw materials which might be necessitated for producing fused magnesium phosphate, if such a sort of fertilizer is chosen.

In case the results of the first stage are encouraging, the Project advances to the second stage which aims at the designing of a mining plan with a pre-feasibility study of the prospect.

The Scope of Work was signed on the 18th day of June, 1984, between the Zambia Industrial & Mining Corporation Limited and the Japan International Cooperation Agency.

1.5 Programme of the Study

The Chilembwe prospect comprises four apatite bodies which are located in the area of syenite mass. The Preparatory Mission selected two targets to be drilled, namely Nos. 2 & 4 Orebodies and allotted eighteen holes of 60 m each. Although nine holes had been sunk at No. 2 Orebody, an additional six holes were recommended to confirm the ore reserves, for two thirds of indicated reserves came from this orebody. No. 4 Orebody had not been drilled yet.

At the outset, the two conditions for advancing to the second stage were laid down.

(1) Ore reserves over 1.5 million tonnes

This condition is based on the assumption that deposits here under review should produce an amount large enough to provide 50% of the total amount of the phosphatic fertilizers consumed in Zambia for 10 to 15 years.

(2) Concentrate Grade over 25% P_2O_5

Dressed ore with a concentrate grade over 20% can be used satisfactorily for the production of fused magnesium phosphate. An improvement of quality requires an increased number of stages in the course of the dressing process, causing an increase in the initial and the running costs. But a high quality of concentrate reduces the required total amount to be transported and thus reduces its transportation cost. The requirement of over 25% was decided taking these conditions into account.

Initially, eighteen drill holes totalling 1,080 m were allocated for phosphate prospecting and three further holes were provided to confirm the existence of magnesium resources. The dressing test was intended to recover a concentrate bigger than 150 meshes in grain size to prevent the concentrates from being blasted off by the gas generated from the carbonates in an electric furnace.

The results of the first stage are summerized in Chapters 2 to 4, and the production plan and the bases of designing are explained in Chapters from 5 to 10.

If the national consumptions of P_2O_5 stands at 20,000 tonnes per annum, and a half of the amount is supplied from the Prospect, the tonnages required per year are $10{,}000 \div (0.115 \times 0.884) = 98{,}400$ where 0.115 is a grade of minable ore and 0.884 is a recovery rate. The dressing plant is designed of a capacity of 104,000 tonnes in 290 working days per annum, with the maximum capacity of 400 tonnes per day. The working days of the mining section 9
are reduced by 30 for expected unworkable days in a rainy season. The mining capacity

Table 1 Drilling Scheme

	INITIAL		ACTUAL	
	NUMBER OF HOLES	LENGTH ALLOTTED	NUMBER OF HOLES	LENGTH DRILLED
No. 2 Orebody	6	360.0 m	10	534.8 m
No. 4 Orebody	12	720.0	10	468.7
Total Phosphate	18	1,080.0	20	1,003.5
Dolomite	3	180.0	5	180.0
TOTAL	21	1,260.0	25	1,183.5

stands at 400 tonnes per day, for 260 days a year. The technical data is attached in the end of a relevant chapter.

This report stands at a level of a pre-feasibility study and is subject to revision prior to implement the Project.

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2. CHILEMBWE PROSPECT

2.1 Location

The Prospect has been named after a village called Chilembwe which is situated some 8 km northwest of the area. The Chilembwe Prospect is in the vicinity of 13°59'S, 31°41'E, and its camp site is located some 28 km north of Sinda. All of the deposits are found within a distance of 9 km to the north of the camp. Sinda is situated 460 km east of Lusaka and both are connected by the paved Great East Road which runs to Malawi.

2.2 Physiography

The area lies roughly 30 km north of a watershed which divides the drainage basins of Zambezi and Luangwa Rivers. The prospect is located at the headwaters of Lusandwa, and four kilometers east of the area, there runs the River Kasangazi from south to north to join the Lusandwa and then the River Luangwa. The Mankwala Dam is situated in the middle part of Kasangazi to the east of the Prospect.

The Prospect lies on a plain, moderately timbered with shrubs, at the western slope of Mkangala Hills, dipping slightly westward at the elevation of 920 m. The average precipitation is about 1,000 mm annually and most of the precipitation falls mainly as brief cloudbursts from November to March.

2.3 Previous Works

The Prospect is covered under Sinda West P.L.183, formerly known as Lusandwa P.L.144. Since 1980, a detailed prospecting programme has been carried out by MINEX, comprising geological, geochemical and geophysical surveys.

A radiometric survey was applied for to obtain moderate but well defined anomalies over the areas of No. 1 and No. 2 Orebodies. A resistivity method was used to identify pegmatite bodies. Magnetic responses were poor and a gravitational survey remains untested due to the unavailability of instruments.

Fifteen holes totalling some 805 m were drilled in 1982 to investigate geophysical anomalies and the downward extension of Nos. 1 to 3 Orebodies. Five holes, from DDH.1 to 5, were drilled at No. 1 Orebody and DDH.13 was put down at No. 3 Orebody. The remaining nine holes were sunk over the area of No. 2 Orebody. Apart from these, an uncountable number of pits and several trenches were excavated.

According to the report prepared by the Preparatory Mission, Orebody No. 1 has dimensions of 65 m long and 35 m wide, reserves of which were estimated at 170,000 tonnes being 15% P_2O_5, to the depth of 40 m from the surface, though a drill hole penetrated ore to a depth of 56 meters. Orebody No. 2 was delineated as 170 m long and 50 m wide, being one million tonnes with an average grade of 12%, to a depth of 60 meters. Orebody No. 3 is 60 m long and 15 m wide. Some 54,000 tonnes of ore were indicated to the depth of 30 m, grading 13%. Although Orebody No. 4 had dimensions of 100 m long and 25 m wide, it had not been drilled yet. From this background, No. 2 and No. 4 were selected to be taken up for additional drilling.

2.4 Geology

Most of the area is underlain by syenite and monzonite rocks which intruded into the Lusandwa Group of the Muva System. The syenites, in association with large masses of granite, form a part of Sinda Batholith. The Lusandwa Group comprises concordant gneisses and associated metasediments in the Chindeni Mobil Belt that trends north-northeast. All of these rocks are of the Pre-Cambrian age.

2.5 Mineral Deposits

The Chilembwe Prospect refers to a number of phosphate deposits which occur in the area of syenite mass. The plutonic rocks range in composition from mica syenite to monzonite. These are holocrystalline and are coarse to medium grained. Mica-rich ultrabasic rocks, with a general preponderance of hornblendite, are recognized in the area of Nos. 3 and 4 Orebodies. Pegmatite is of the latest generation and consists of large alkali-feldspar and quartz crystals.

Phosphate deposits can be apparently classified into two types based upon their modes of occurrence. Both the Orebodies Nos. 1 and 2 occur as massive leucocratic bodies composed of apatite with association of quartz, whereas No. 3 and No. 4 Orebodies comprise apatite, mica, amphibole and pyroxene and have the forms of sills or dikes, in association with the fine grained ultrabasic rocks which intruded into the syenite mass.

The No. 2 Orebody occurs elliptically along a small hill and floats of ore are widely distributed at 190 meters north-south in length and 60 meters in an east-west direction. The south-eastern corner of the orebody appears to have been cut out by pegmatite. The adjacent area to the south has no prospects for the discovery of orebodies. Limonitic gossanous rocks are widespread along a small valley which confines the southern borders of the ore floats. The mineral deposit appears to be a mass derived from late-magmatic segregation of alkali igneous rocks.

The No. 4 Orebody occurs within a small flat area. After the visit of the Preparatory Mission, the existing tenches were further extended to the east and two trenches were added. Samples taken from these trenches delineated a zone of apatite mineralization in the weathered ultrabasic rocks over an area of 100 m long and 40 m wide. A hill on the east is occupied by outcrops of gossanous monzonite and fresh pegmatite with sporadic hornblendite.

2.6 Drilling

Tenders were invited in March, 1984, to undertake the drilling programme and a Romanian company, named Geomin, was selected and recommended by MINEX. Some difficulties were experienced during negotiations with this contractor, because of monetary problems such as inflation and devaluation. Drilling operation started on the 28th of June

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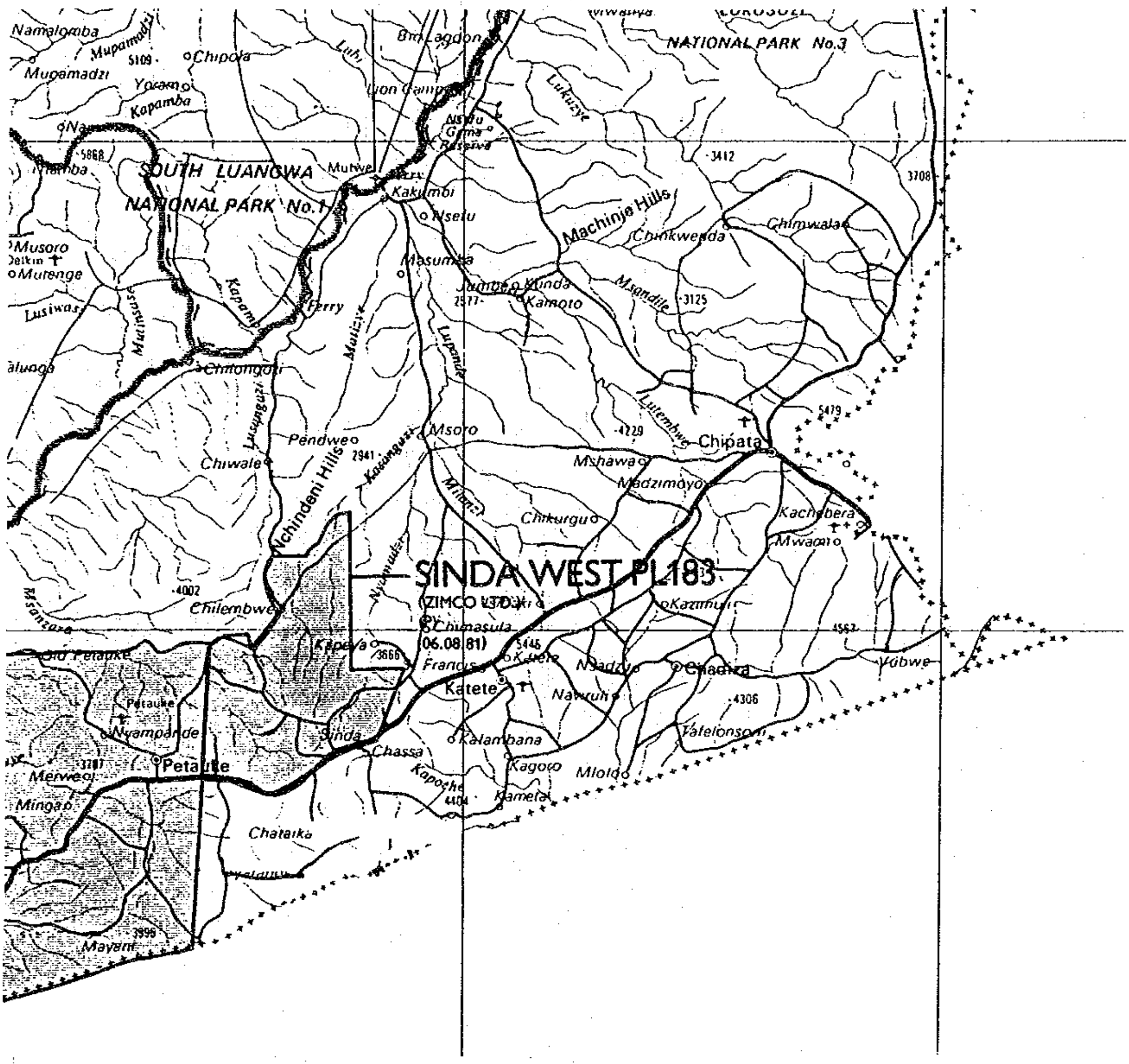

Kilometres 15 10 5 0 15 30 45 60 75 90 105 120 135 150 165 Kilometres

Fig. 2-1 **PROSPECTING, EXPLORATION AND MINING LICENCE AREAS FEBRUARY, 1982**

PL PROSPECTING LICENCE
(WITH DATE OF ISSUE)

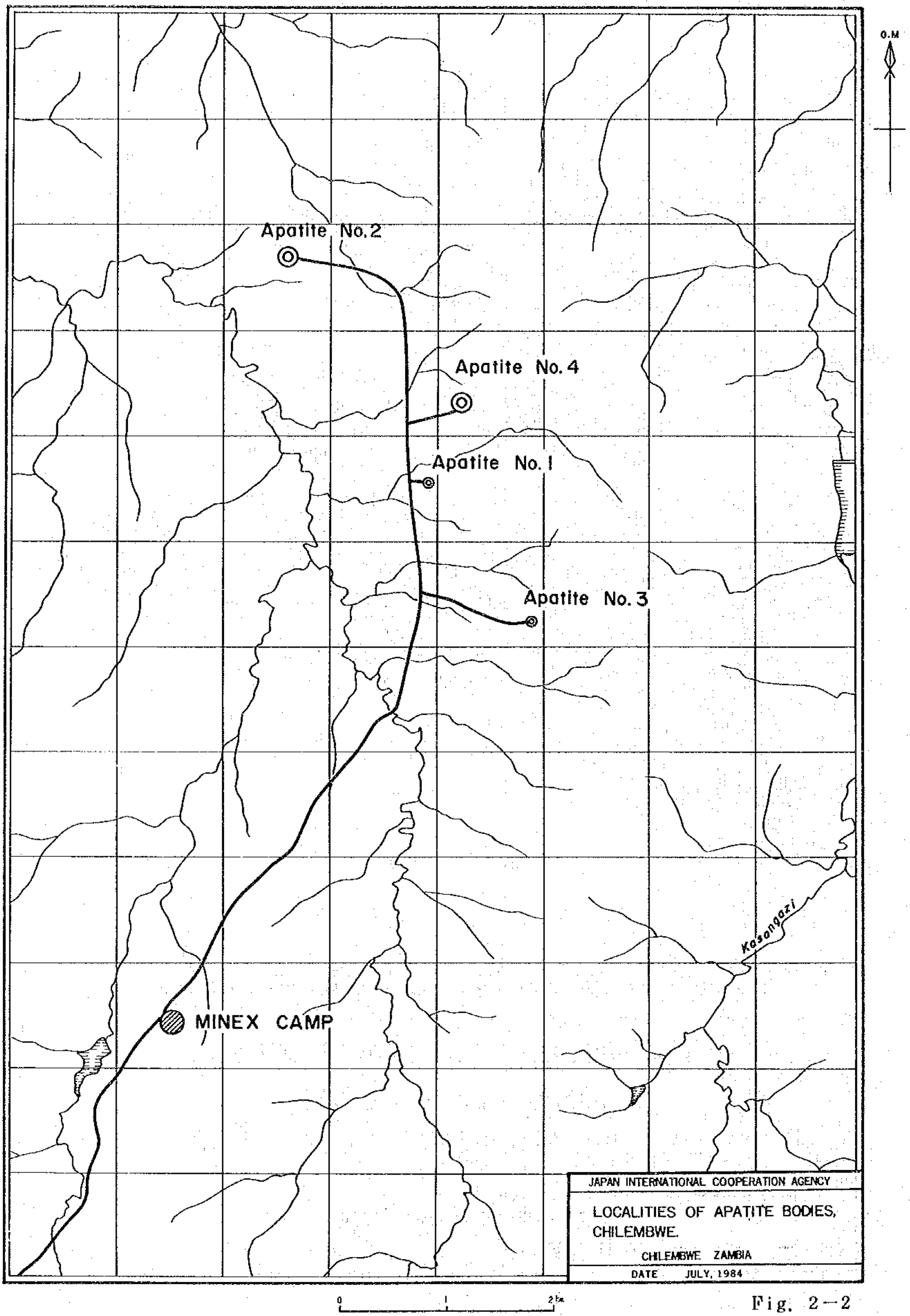
G.M
Apatite No.2
Apatite No.4
Apatite No.1
Apatite No.3
Kasongazi
MINEX CAMP
JAPAN INTERNATIONAL COOPERATION AGENCY
LOCALITIES OF APATITE BODIES, CHILEMBWE.
CHILEMBWE ZAMBIA
DATE JULY, 1984
0
1
2km

Fig. 2—2

and ended on the 8th of August.

2.6.1 No. 2 Orebody

The deepest mineralization of the No. 2 Orebody was recorded in the inclined DDH.8 at a calculated vertical depth of 78.4 m from the mouth. But the present target is restricted to a depth of sixty meters. Eight spots were selected to be drilled, most of them were placed outside the previous delineation with anticipation of finding an additional quantity of ores. In the course of the drilling operation, the best result was obtained from Hole II-5 to the west of the initial area. Subsequently, two holes were added to confirm the western margin of the orebody.

The syenites are coarse to medium grained holocrystalline rocks composed of mainly alkali-feldspar, biotite, amphibole and a small amount of pyroxene. The rocks form a wall rock and are recognized as xenolith in the orebody.

Hornblendite occurs as thin dikes or sills, mainly around the peripheries of the orebody. Its width does not normally exceed one meter. The rock is impregnated with a small amount of apatite and is also cut by veinlets of apatite rocks.

Apatite rock is a medium grained homogeneous rock comprised of apatite and quartz with subordinate amounts of feldspar, biotite and amphibole. Apatite body often involves syenite and hornblendite as xenoliths and is cut by pegmatite dykes of several meters in width.

2.6.2 No. 4 Orebody

Initially, twelve spots were pegged at the No. 4 Orebody, but two spots at the northern end were abandoned. From observation of Trenches 1 and 2, mineralization appears to thin out towards the northwest. Other holes were rather broadly rearranged to cover the south eastern area, with expectations that ore might be widespread in this direction.

Apatite rock is in the form of layers in flat lying sills of ultrabasic rocks which occur in syenite mass. Two sheets of sills with a minable thickness have been penetrated. Ultrabasic rocks are comprised of feldspar, biotite, hornblende and pyroxene. The grain size of these minerals ranges from less than one to ten millimeters. Compared with the geology of No. 2 Orebody, this type of ore seems to be a product of an earlier stage of segregation.

2.7 Chemical Analysis

Samples were assayed for P_2O_5 by MINEX, using a colorimetric method at the field laboratory in the camp. As shown in the histogram, two contrasting patterns of assay value distribution were obtained. Around seventy percent of the samples have a yield value of less than 6% content. A cut-off grade of 6% was tentatively applied for calculation of ore reserves.

From these samples, forty seven were re-assayed in Japan and the following correlation coefficient and regression equation were obtained.

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$$r = 0.98$$
$$Y = 1.29X - 0.59$$
$$\sqrt{V} = 2.11$$

where

r ..... the correlation coefficient
Y ..... assay value obtained in the camp
X ..... assay value obtained in Tokyo
V ..... unbiased estimation of covariance

2.8 Ore Reserves

Tabulated are ore intersections, each of which has an interval of more than one meter thick and grades over 6% P_2O_5. On the assumption that each intersection has a stretch, at least equal to its interval of ore intersection, the preliminary outlines of the orebodies are determined. These outlines are then revised by data which are available from previous drill records and surface geology.

The area of influence of any drill hole is considered to be a polygon bounded midway to all surrounding holes. All vertical holes including previous drilling are taken into present calculation, but the records of inclined drill holes are excluded except in case where information of the deep zone is not covered by the vertical holes. The DDH.6 and DDH.11 had been stopped at shallow depths. Compared with the neighbouring areas, the ore intersections of 20 and 9.5 m were added to the bottoms of these holes respectively, using the mean assay value of each hole as marked by the asterisks.

Apart from the main intersection of each drill hole, an isolated ore section more than one meter thick is deemed to be recoverable if it exists within a frustum which should be excavated during the course of preproduction of the main orebody. These are grouped in miscellaneous ores. In this case, an angle of 45 degrees is applied for a pit slope.

As the results of the present investigation, 1.6 million tonnes of ore grading 11.8% P_2O_5 have been estimated as follows:

	Tonnage	Grade	Contents
	1,000 t	$\%P_2O_5$	1,000 t
No. 2 Orebody	1,421	12.1	172
No. 4 Orebody, shallow horizon	107	10.3	11
ditto , deep horizon	113	9.5	10
Total	1,641	11.8	193

It is evident that the ores in the deep horizon at the No. 4 Orebody involves a high stripping ratio. Mining of such ores should be avoided as much as possible, from the economical point of view. No. 3 Orebody was also avoided for the same reason. But, if the ores in the deep horizon of No. 4 Orebody can be substituted by the ores of No. 1 Orebody, the total ore reserves of the Chilembwe Prospect will be expressed as follows:

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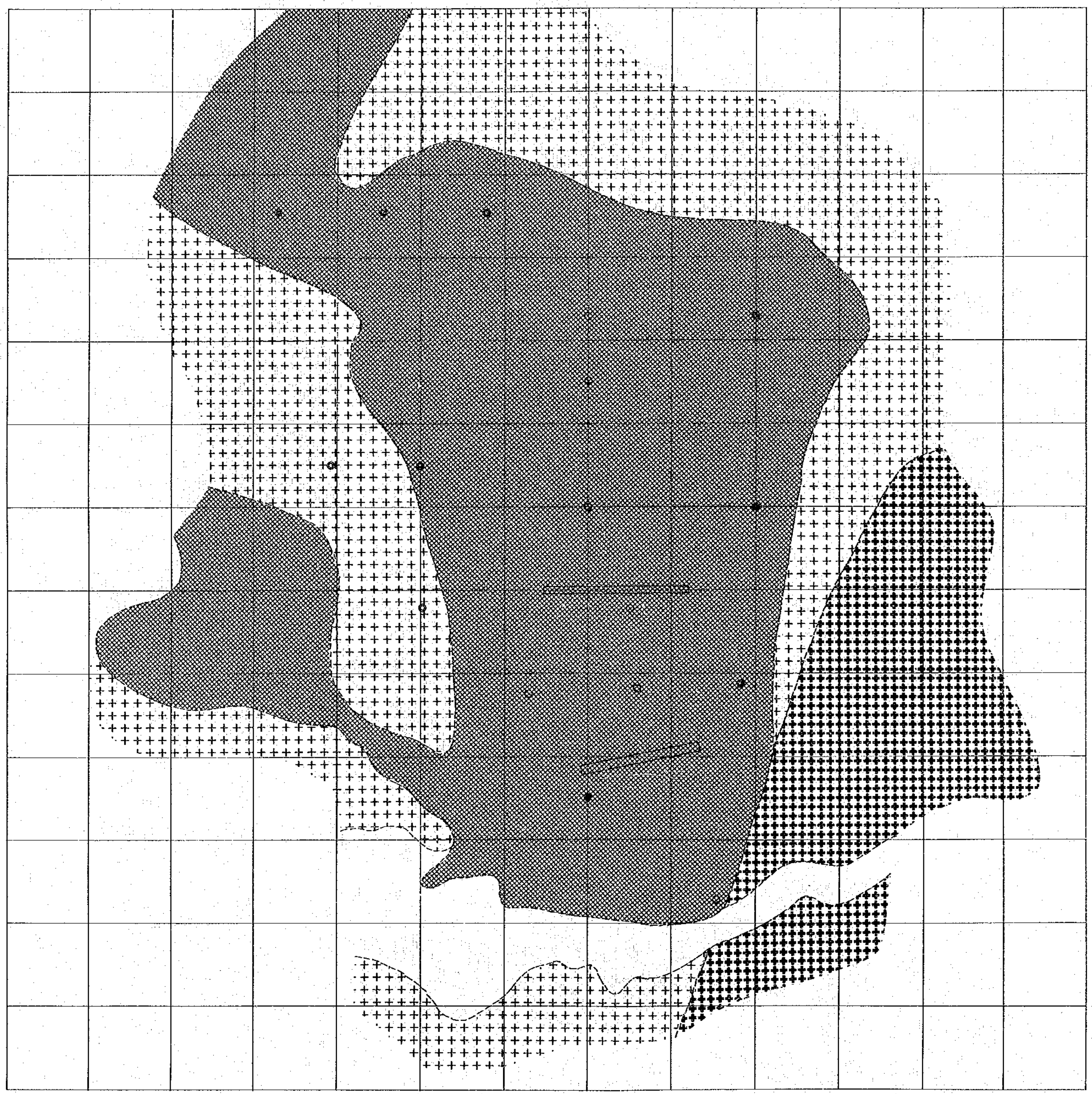


Legend

Pegmatite

Apatite rock

Syenite ~ Monzonite

JAPAN INTERNATIONAL COOPERATION AGENCY

INFERRED GEOLOGY
No.2 OREBODY, CHILEMBWE
CHILEMBWE ZAMBIA

SCALE	1:1,000
DATE	JULY, 1984

Fig. 2-3

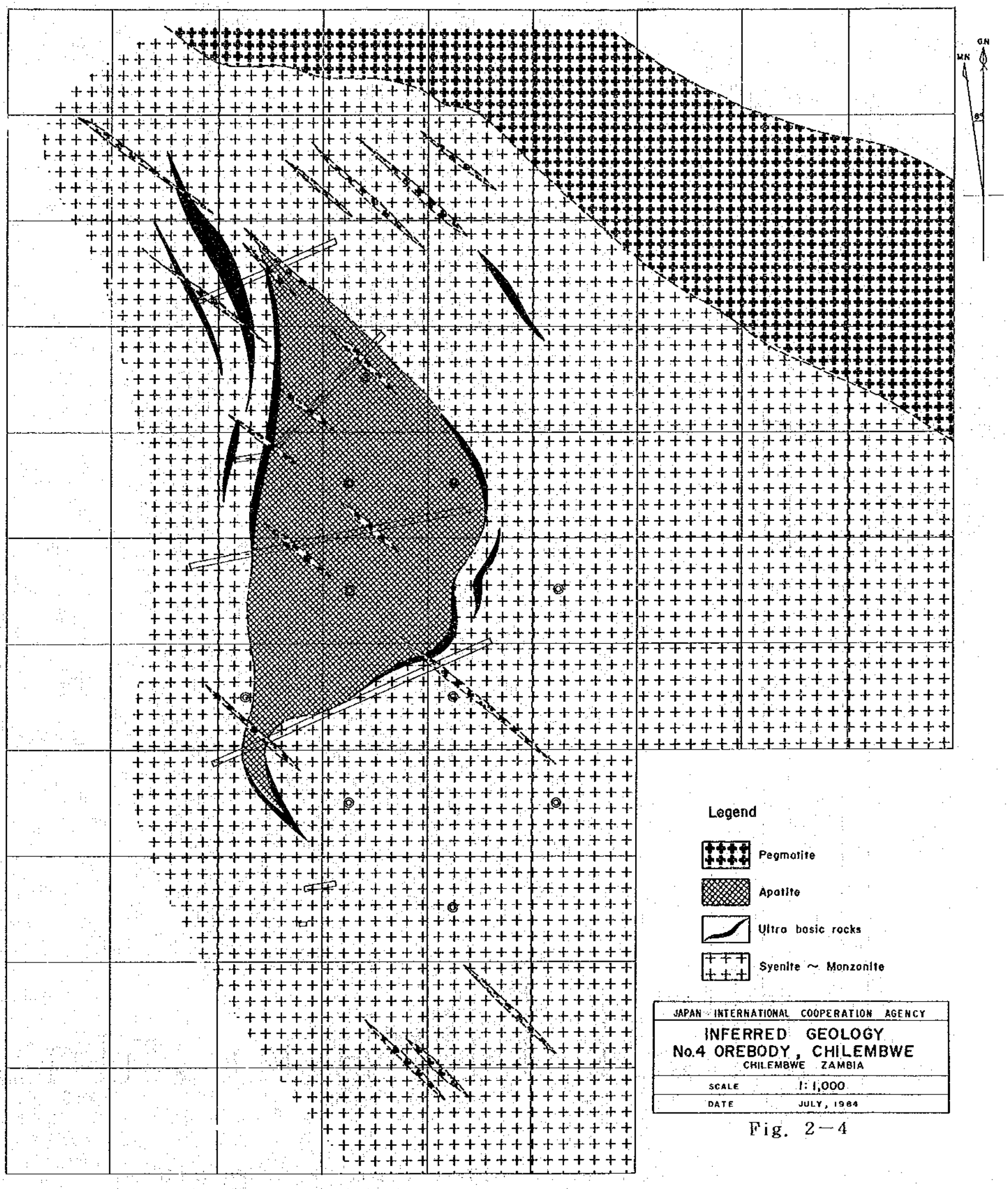

GN
MN
Legend
Pegmatite
Apatite
Ultra basic rocks
Syenite ~ Monzonite
JAPAN INTERNATIONAL COOPERATION AGENCY
INFERRED GEOLOGY
No.4 OREBODY, CHILEMBWE
CHILEMBWE ZAMBIA
SCALE 1:1,000
DATE JULY, 1984

Fig. 2-4

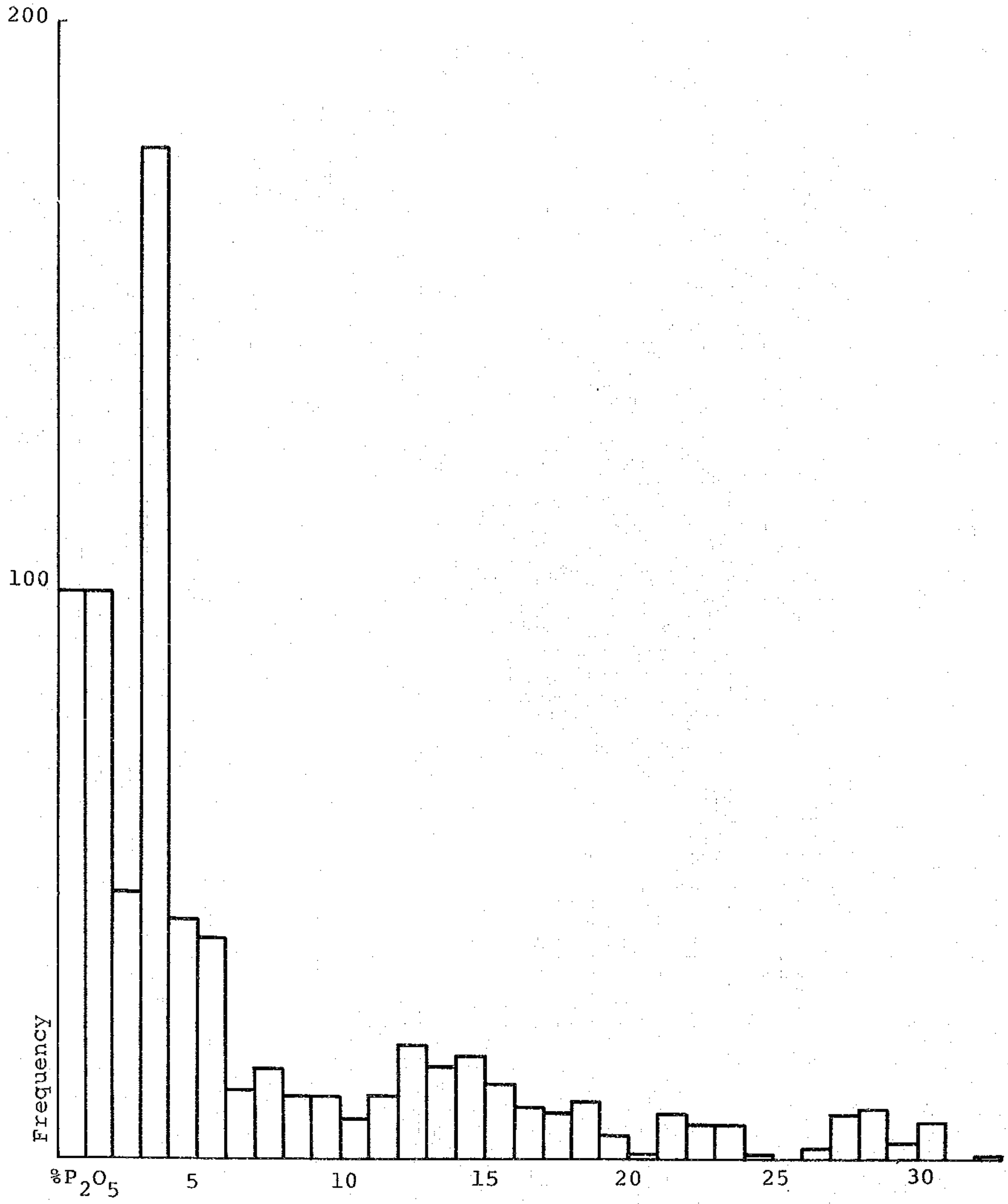


DIAGRAM : DISTRIBUTION OF ASSAY VALUES

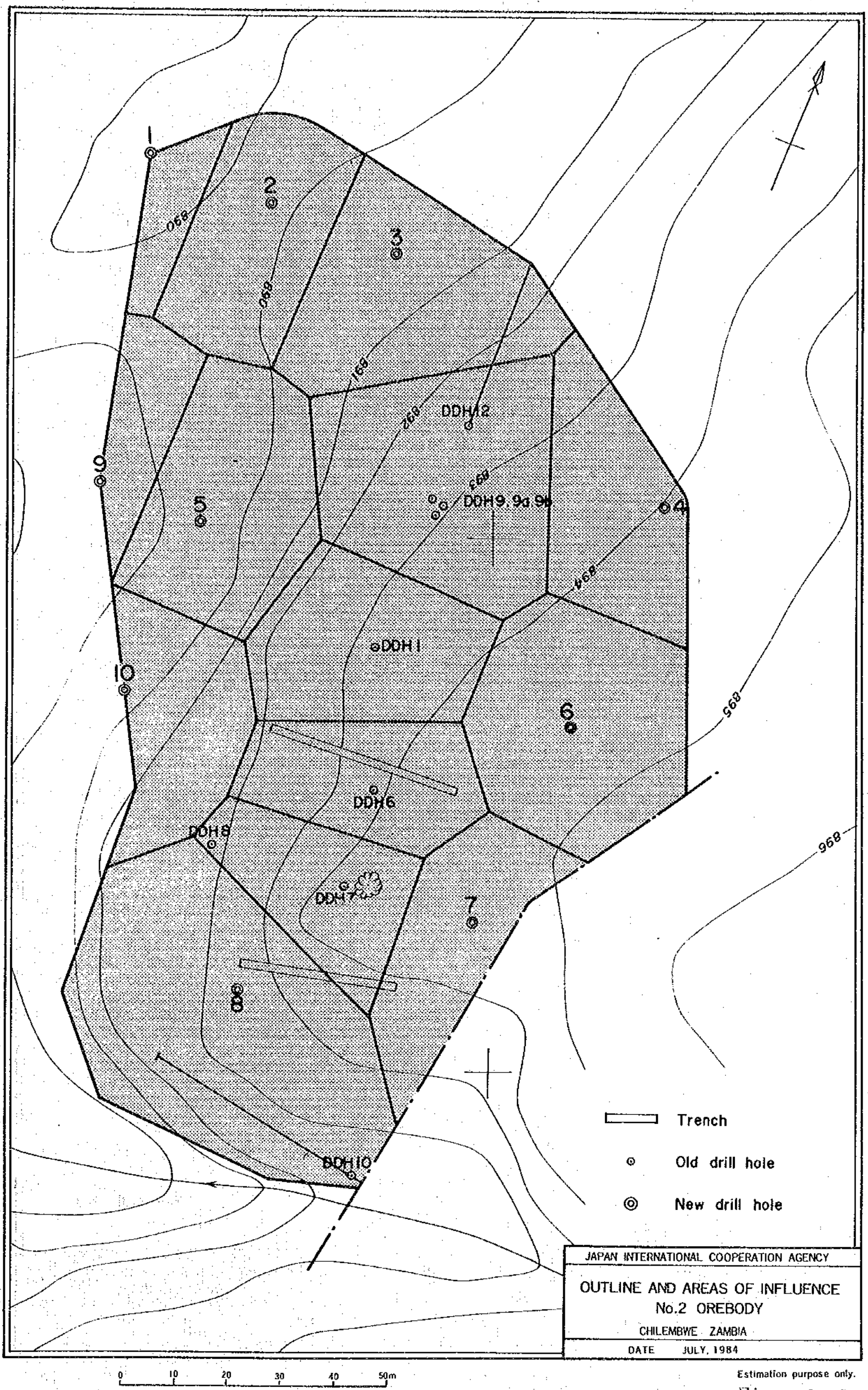
2C

Table 2-1. Drill Hole Intersections: No. 2 Orebody

Hole No.	From m	To m	Thickness m	P_2O_5 %
II-1	18.27	19.63	1.36	10.00
	38.85	40.35	1.50	13.50
II-2	5.00	6.00	1.00	7.00
	6.95	24.28	17.33	11.93
	47.05	48.81	1.76	15.07
II-3	26.06	27.33	1.27	16.00
	38.89	40.03	1.14	8.00
	43.58	45.25	1.67	9.02
	55.37	57.70	2.33	9.00
II-4	14.25	17.00	2.75	7.72
	25.70	30.00	4.30	10.28
	37.55	38.98	1.43	8.00
II-5	7.45	21.00	13.55	14.71
	29.20	30.64	1.44	12.00
	34.10	60.00	25.90	25.49
II-6	0.50	2.50	2.00	6.50
	11.13	33.00	21.87	11.13
	35.75	37.44	1.69	11.00
II-7	5.97	52.00	46.03	12.46
II-8	9.00	10.00	1.00	7.00
	20.45	39.85	19.40	8.30
II-9	21.00	22.80	1.80	6.00
	46.60	53.18	6.58	7.26
II-10	33.00	34.50	1.50	19.00
	40.00	42.40	2.40	8.83

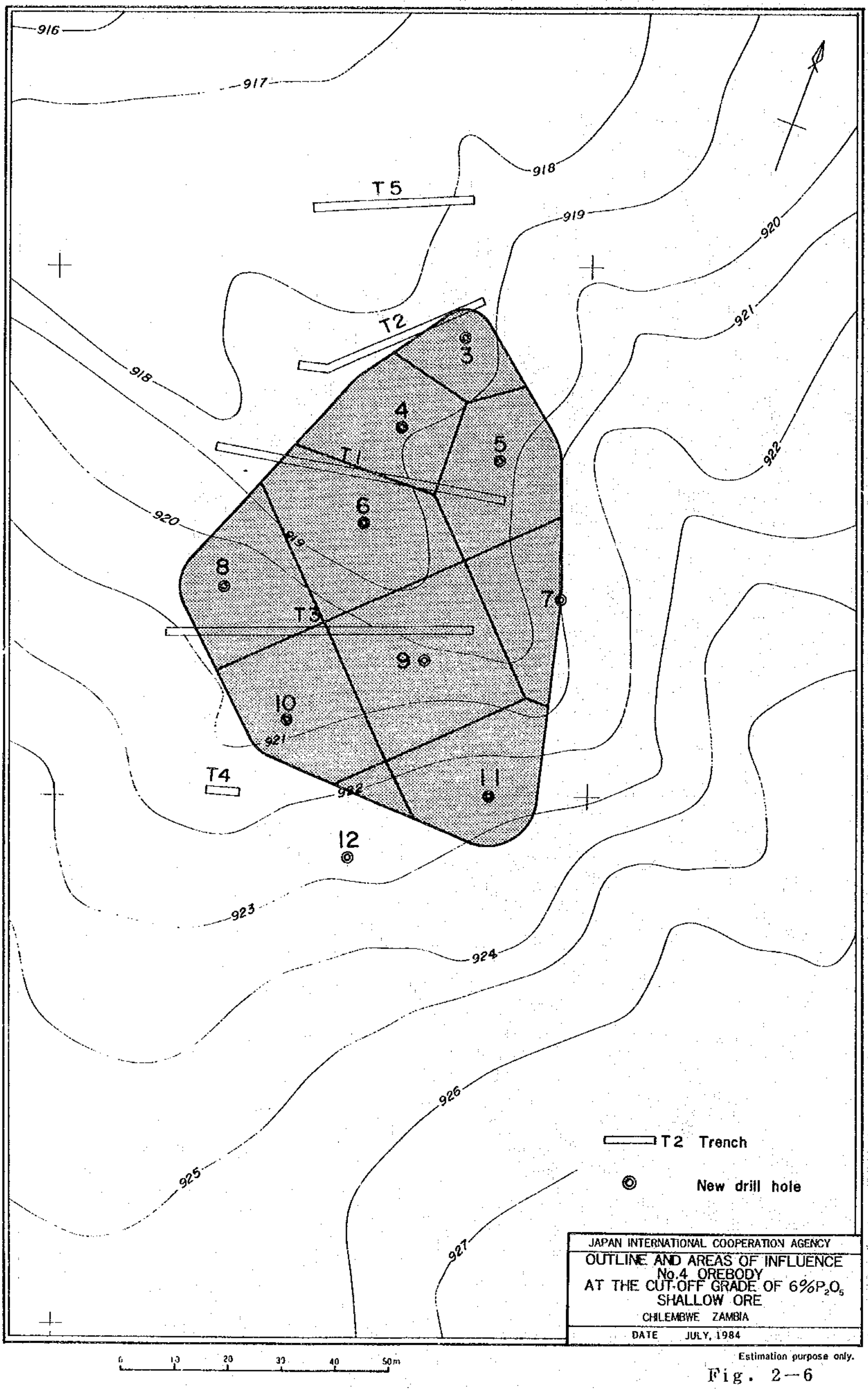
Table 2-2. **Drill Hole Intersections: No. 4 Orebody**

Hole No.	From m	To m	Thickness m	P_2O_5 %
IV-3	1.00	6.00	5.00	8.16
	15.50	16.50	1.00	11.00
	19.60	41.42	21.82	10.80
IV-4	0.00	12.65	12.65	14.75
	17.00	18.00	1.00	15.00
	42.50	49.15	6.65	13.91
IV-5	4.50	16.70	12.20	9.68
	23.97	42.70	18.73	6.60
IV-6	0.06	4.00	3.94	11.37
	7.47	11.77	4.30	8.76
IV-7	55.40	56.60	1.20	13.00
IV-8	3.00	4.00	1.00	7.00
	6.43	14.76	8.33	11.97
	44.00	46.08	2.08	13.00
IV-9	1.20	2.20	1.00	7.00
	6.62	7.62	1.00	7.50
	44.00	45.00	1.00	11.80
IV-10	7.93	9.41	1.48	19.00
	20.50	27.00	6.50	8.92
IV-11	12.20	21.78	9.58	11.06
	54.17	55.17	1.00	11.75
IV-12	41.51	44.70	3.19	6.00



Estimation purpose only.

Fig. 2-5



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T2 Trench
New drill hole
JAPAN INTERNATIONAL COOPERATION AGENCY
OUTLINE AND AREAS OF INFLUENCE
No.4 OREBODY
AT THE CUT-OFF GRADE OF 6% P_2O_5
SHALLOW ORE
CHILEMBWE ZAMBIA
DATE JULY, 1984
0 10 20 30 40 50m
Estimation purpose only.

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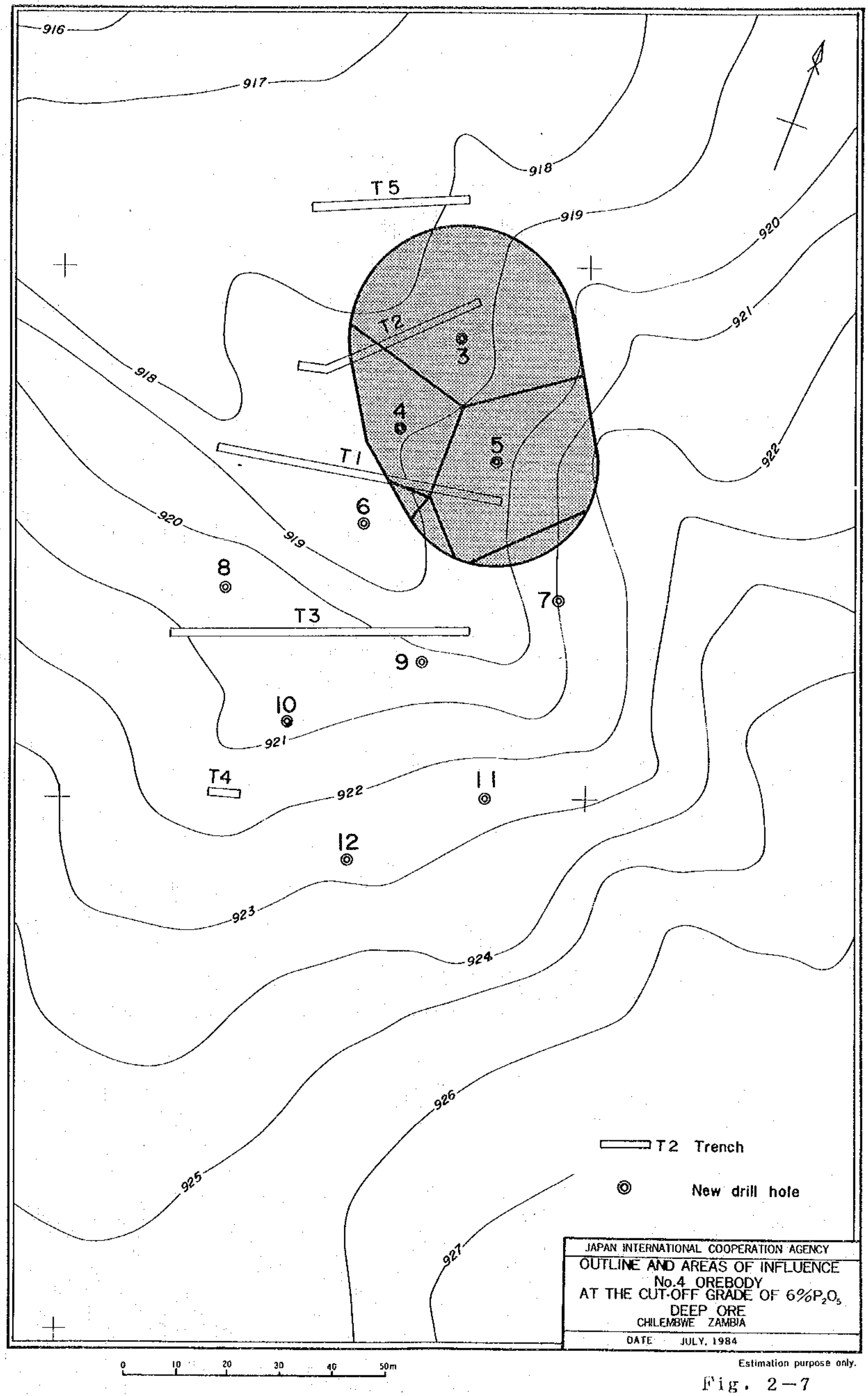
Fig. 2—6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New drill hole
JAPAN INTERNATIONAL COOPERATION AGENCY
OUTLINE AND AREAS OF INFLUENCE
No.4 OREBODY
AT THE CUT-OFF GRADE OF 6%P2O5
DEEP ORE
CHILEMBWE ZAMBIA
DATE JULY, 1984
0 10 20 30 40 50m
Estimation purpose only.

Fig. 2−7

Table 2-3. Miscellaneous Ores : No. 2 Orebody

Hole No.	Area	From	To	Height	Volume	S.G.	Tonnage	P_2O_5	Contents
	m^2	m	m	m	m^3		tonne	%	
II-1	195	18.27	19.63	1.36	265	3.0	795	10.00	79.5
	88	38.85	40.35	1.50	132	3.0	396	13.50	53.4
II-2	352	47.05	48.81	1.76	619	3.0	1,857	15.07	279.8
II-3	867	26.06	27.33	1.27	1,101	3.0	3,303	16.00	529.0
	970	38.89	40.03	1.14	1,105	3.0	3,315	8.00	265.2
	757	43.58	45.25	1.67	1,264	3.0	3,792	9.02	342.0
II-4	992	14.25	15.25	1.00	992	3.0	2,976	9.00	267.8
	845	37.55	38.98	1.43	1,208	3.0	3,624	8.00	289.9
II-6	1,561	0.50	2.50	2.00	3,122	3.0	9,366	6.50	609.0
	931	35.75	37.44	1.69	1,573	3.0	4,719	11.00	519.0
II-9	648	21.00	22.80	1.80	1,166	3.0	3,498	6.00	209.8
II-10	781	33.00	34.50	1.50	1,171	3.0	3,513	19.00	667.4
	438	40.00	42.40	2.40	1,051	3.0	3,153	8.83	278.4
Total					14,769		44,307	9.90	4,390.2

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Table 2-4. Ore Reserve Calculation : No. 2 Orebody

Hole No.	Area	From	To	Height	Volume	S.G.	Tonnage	P_2O_5	Contents
	m^2	m	m	m	m^3		tonne	%	tonne
II-2	1,098	6.95	24.28	17.33	19,028	3.0	57,084	11.93	6,810.1
DDH9	1,716	0.00	62.87	62.87	107,884	3.0	323,652	13.26	42,916.2
II-4	992	25.70	30.00	4.30	4,265	3.0	12,795	10.28	1,315.3
II-5	1,291	7.45	60.00+	52.55	67,842	3.0	203,526	16.89	34,375.5
DDH11	1,032	8.05	40.50+	32.45	33,488	3.0	100,464	9.32	9,363.2
II-6	1,561	11.13	33.00	21.87	34,139	3.0	102,417	11.13	11,399.0
DDH6	918	0.00	25.00	25.00	22,950	3.0	68,850	13.16	9,060.6
DDH7	838	0.00	45.70	45.70	38,296	3.0	114,888	9.83	11,293.4
II-7	925	5.97	52.00	46.03	42,577	3.0	127,731	12.46	15,915.2
II-8	2,542	20.45	39.85	19.40	49,314	3.0	147,942	8.30	12,279.1
II-9	648	46.60	53.18+	6.58	4,263	3.0	12,789	7.26	928.4
Subtotal	13,561			31.26	424,046		1,272,138	12.23	155,656.0
Miscell.		see attached sheet			14,769		44,307	9.90	4,390.2
Total					438,815		1,316,445	12.15	160,046.2
DDH12	380	14.77	32.80+	18.03	6,851	3.0	20,553	8.02	1,648.3
DDH11	1,032	40.50	50.00	9.50	9,804	3.0	29,412	9.32*	2,741.1
DDH6	918	25.00	45.00	20.00	18,360	3.0	55,080	13.16*	7,248.5
Total					35,015		105,045	11.07	11,637.9
Grand Total					473,830		1,421,490	12.07	171,684.1

Table 2-5. Ore Reserve Calculation : No. 4 Orebody

Hole No.	Area m^2	From m	To m	Height m	Volume m^3	S.G.	Tonnage tonne	P_2O_5 %	Contents tonne
IV-3	236	1.00	6.00	5.00	1,180	3.0	3,540	8.16	288.8
IV-4	447	0.00	12.65	12.65	5,654	3.0	16,962	14.75	2,501.8
IV-5	532	4.50	16.70	12.20	6,490	3.0	19,470	11.86	2,309.2
IV-6	743	0.06	11.77	11.71	8,700	3.0	26,100	7.22	1,884.4
IV-8	512	6.43	14.76	8.33	4,264	3.0	12,792	11.97	1,531.2
IV-9	800	1.20	2.20	1.00	800	3.0	2,400	7.00	168.0
IV-10	504	2.25	9.41	7.16	3,608	3.0	10,824	6.80	736.0
IV-11	507	12.20	21.78	9.58	4,857	3.0	14,571	11.06	1,611.5
Shallow Ore	4,281			8.30	35,553		106,659	10.34	11,030.9
IV-3	1,013	19.60	41.42	21.82	22,103	3.0	66,309	10.82	7,174.6
IV-4	329	42.50	49.15	6.65	2,187	3.0	6,561	13.92	913.2
IV-5	712	23.97	42.70	18.73	13,335	3.0	40,005	6.60	2,640.3
Deep Ore	2,054			18.31	37,625		112,875	9.50	10,728.1
Total					73,178		219,534	9.91	21,759.0

	Tonnage 1,000 t	Grade $\%P_2O_5$	Contents 1,000 t
No. 1 Orebody	170	15.1	25
No. 2 Orebody	1,421	12.1	172
No. 4 Orebody	107	10.3	11
Total	1,698	12.3	208

Several parts remain undrilled due to the limitation of the initially specified number of drill holes.

The northern end of DDH.12 is in a zone of 23% P_2O_5 at the calculated depth of 32.8 m and its extension is still open.

Work to date in the form of geological mapping has indicated further extended mineralization towards the northwest. Floats consist mainly of apatite rocks but their distribution is rather scarce.

A small hill at the west of Hole II-10 has a dimension of sixty meters in diameter and is overlain by a layer of apatite floats. This area indicates the presence of a possible satellite orebody.

The writers are of the opinion that these areas can be excavated during the course of preproduction stripping and no additional drilling is recommended at this stage.

Apart from these adjacent areas, another prospect named Chakanga has been reported to have a potential of 1.15 million tonnes of ore being more than 4% P_2O_5. The prospect is situated 12 km west of the camp and apatite is accompanied with biotite-bearing hornblende monzonite to hornblendite.

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3. MAGNESIUM PROSPECT

3.1 Magnesium Resources

The fertilizer industry of fused magnesium phosphate consumes an apatite concentrate and serpentinite. A silicified serpentinite body is reported to be located to the west of Muloba, a village located about 40 km northeast of Kapiri Mposhi. Two localities of basic intrusives are known to be on the northern bank of the Zambezi River, 120 km southeast of Lusaka. But the magnesium contents of these rocks are not known.

Dolomite can be substituted for magnesium resources, hence attention is given to the dolomites in the Basement Complex located north of Petauke and to the Lusaka Dolomites of the Katanga System. In the Lusaka district, several occurrences of dolomite have been recorded in the Cheta Formation and the overlying Lusaka Dolomite.

3.2 Scheme of Investigation

The purpose of the present investigation is to provide basic information for full-scale prospecting of magnesium raw materials which will be required in case the choice of fused magnesium phosphate is made.

A reconnaissance survey was carried out by MINEX and a prospect named Kyindu Ranch has been taken up. The tentative drilling lengths of 180 m, being 3 holes of 60 m each, were allotted in the Scope of Work.

3.3 Kyindu Ranch Prospect

The prospect is in the vicinity of 15°32'S and 28°31'E, at an elevation of about 1,300 m. Part of this plateau is utilized for commercial farming. The prospect refers to a deposit at the site of Kyindu Ranch, a cattle station some 27 km southeast of Lusaka. A graded road runs across the plateau to Leopards Hill and the prospect lies some 5 km south of the road. The Kyindu Ranch is situated at the headwaters to the Mkwisi, a tributary of the Zembezi. Drainage is from the north to the south running parallel and perpendicular to the strike of the Katanga metasediments that runs north westerly.

The Basement Complex of the area mainly comprises gneisses and is overlain by the Cheta Formation which consists of a sequence of psammitic and pelitic schists, banded limestone and pelitic schist with quartzite, in an ascending order.

Overlying the rocks of the Cheta Formation is a synclinorial belt of dolomites and dolomitic limestones of the Lusaka Dolomite. The Lusaka Dolomite forms a belt of 5 km in width, running west north west. This trend twists in the direction of northwest at the eastern margin of the Lusaka Dolomite where the prospect is situated.

The area is underlain by a thick sequence of grey and white limestone. A dolomite bed crops out elliptically with a width of 100 m in the north-south direction. Dolomite is pale yellowish brown to white or pinkish in places. Often the crystals exceed several centimeters in grain size but most dolomite rocks are massive and fine grained.

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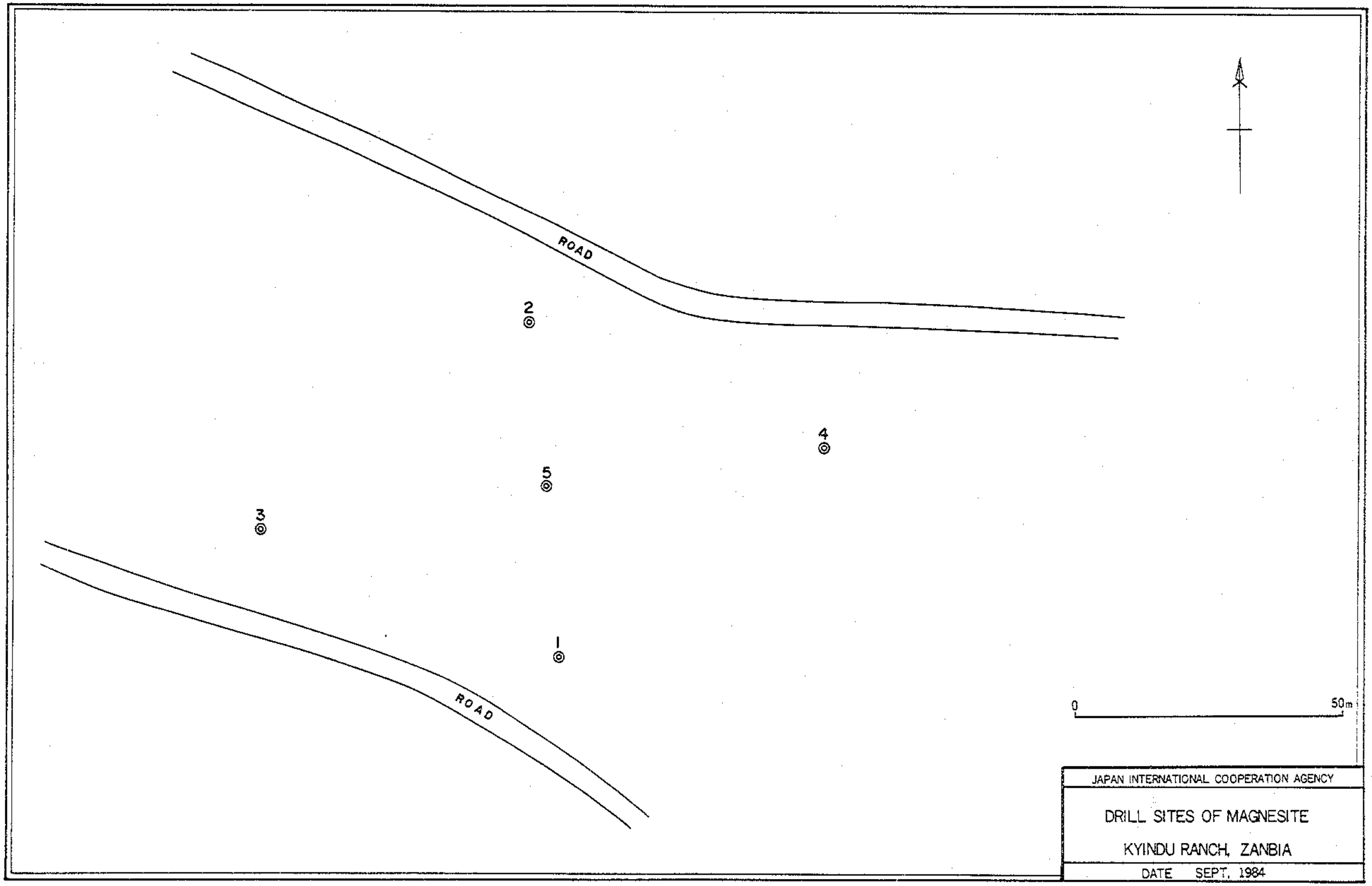
ROAD
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1
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50m
JAPAN INTERNATIONAL COOPERATION AGENCY
DRILL SITES OF MAGNESITE
KYINDU RANCH, ZANBIA
DATE SEPT. 1984

Fig. 3

Table 3-1. Comparison of Assay Results

Laboratory			Lusaka		Tokyo	
Hole No.	From	To	CaO	MgO	CaO	MgO
	m	m	%	%	%	%
2	9.85	10.85	29.66	20.31	27.64	15.84
3	34.30	35.30	30.89	21.42	28.42	16.53
3	35.30	36.30	30.15	20.46	28.32	16.40
4	10.11	10.30	30.15	21.87	28.62	16.71
4	10.30	11.30	30.04	24.19	28.74	16.93
5	10.00	11.00	30.61	21.41	28.43	16.50
5	11.00	12.00	29.54	27.05	28.29	16.72
	Mean		30.15	22.39	28.35	16.52

Table 3-2. Dolomite Intersections

Hole No.	From	To	Run	$MgCO_3$
	m	m	m	%
1	3.75	11.17	7.42	35.85
2	6.25	15.55	9.30	36.67
3	4.00	10.28	6.28	30.18
4	0.00	21.90	21.90	41.37
5	3.00	19.00	16.00	40.25

3.4 Drilling

Drilling operation was undertaken by GEOMIN. Due to the flatness of the area, the depths of each hole were reduced. Four holes were placed at the vertices of equilateral triangles spacing sixty meters each and an additional hole was put down at the center of the area by MINEX to confirm the mode of occurrence and the continuity. Consequently, five holes with a cumulative depth of 180 m were drilled.

3.5 Chemical Analysis

All cores were assayed for calcium and magnesium by the MINEX laboratory in Lusaka and from these, seven samples were re-assayed in Japan. As the asssay values from the Lusaka laboratory were expressed in terms of elements, the data was converted into the form of oxides. Compared with the values obtained in Tokyo, the figures reported in Lusaka are somewhat higher, especially in the case of magnesium oxide. The excess amounts may be observed when these figures are converted into the form of carbonates.

3.6 Results of Drilling

All holes were drilled in the area of dolomite occurrences but encountered limestone at shallow depths in the western part. The dolomite bed seems to thicken in the northeast. The dolomite intersections are tabled with the mean values, using the data reported from the Lusaka laboratory.

If the area of estimation is restricted within a rhomb surrounded by these holes, reserves of some 125,000 tonnes of dolomite being 39% $MgCO_3$ can be obtained. But this area forms only a part of the dolomite distribution and the tonnage figure indicated is no more than the minimum amount calculated from the present drilling. The results do not indicate more than the fact that the existence of dolomite reserves has been confirmed.

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4. CONCENTRATION TEST

4.1 Sample

Sampling for a dressing test was done by hammering the bottoms and walls of the trenches after the surface soil was removed. All samples were taken from trenches of No. 2 and No. 4 Orebodies, totalling some 80 kg, and sent to Japan by air for laboratory experimentation.

The result of the chemical analyses of these samples is given as follows.

Chemical Assay of Samples

	P_2O_5 %	Al_2O_3 %	SiO_2 %	Fe_2O_3 %
No. 2 Orebody	21.56	1.99	40.01	1.86
No. 4 Orebody	18.88	2.94	28.39	6.88
Weathered Ore	12.47	5.85	43.50	6.21

4.2 Mineralogical Examination of Samples

4.2.1 Chemical analyses and crushing characteristics

A feed sample for concentration tests was prepared by crushing it to a minus 14 mesh with a Dodge crusher and a crushing roll. A sample for the Work Index measurement was crushed to a minus 6 mesh.

Plotting the size distribution of the feed samples on the Rosin-Rammler-Bennett's diagram, the representative particle size "d" does not differ substantially among the three samples, and inclination "n" is almost identical. It seems that the three samples have the same size distribution, meaning that they have the same crushing characteristics.

Phosphorus grades of each size fraction are given in Table 4-1.

4.2.2 X-ray diffraction analysis

The following three groups of samples were tested by X-ray diffraction.

a) Samples from Nos. 1, 2, 3 and 4 Orebodies.

b) Magnetic particles of phosphate concentrate. Some differences are apparent with the kind of gangue minerals: quartz prevails in Nos. 1 and 2 Orebodies, and amphibole and pyroxene in Nos. 3 and 4 Orebodies.

The phosphate mineral of the concentrate was identified as hydroxyl apatite $Ca_5(PO_4)_3OH$.

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Table 4-1 P_2O_5 Distribution by Size Fraction of Feed Sample

Mesh	No. 2 Orebody		No. 4 Orebody		Weathered ore	
	Wt %	% P_2O_5	Wt %	% P_2O_5	Wt %	% P_2O_5
+ 20	14.5	17.78	18.1	12.16	16.1	10.03
24	7.9	20.80	7.8	13.94	7.9	11.76
28	8.8	21.77	8.6	16.04	8.2	13.63
35	17.0	23.99	13.4	20.19	14.9	16.35
48	14.5	23.24	12.9	23.82	13.3	18.33
65	11.0	22.28	10.7	24.14	10.8	19.35
100	8.4	21.61	9.0	23.82	8.6	18.34
150	5.7	21.28	5.6	22.14	5.4	17.13
–150	12.2	17.94	13.9	14.59	14.8	9.84
Total	100.0	21.26	100.0	18.22	100.0	14.58

4.2.3 The result of the chemical analysis of phosphate concentrate

Assay values of phosphate concentrate are as follows:

P_2O_5	T.Fe	Fe^{2+}	S	CaO	MgO
34.8%	0.90%	0.28%	$<$0.01%	48.3%	0.93%

Al_2O_3	SiO_2	Na_2O	K_2O	CO_2	F	Cl
0.74%	12.40%	0.22%	0.1%	$<$0.01%	0.78%	0.80%

4.3 Heavy-media Separation

The coarse size fraction of the feed sample (–14/+20 mesh) was treated with acetylens tetrabromide (Sp. Gr. 2.95).

In the case of No. 2 Orebody, a high grade concentrate (P_2O_5 = 36%) is obtained with a low phosphrous recovery (60%). On the other hand, a high recovery was achieved from the No. 4 Orebody and weathered ore, but the concentrate grades are very low.

4.4 Flotation

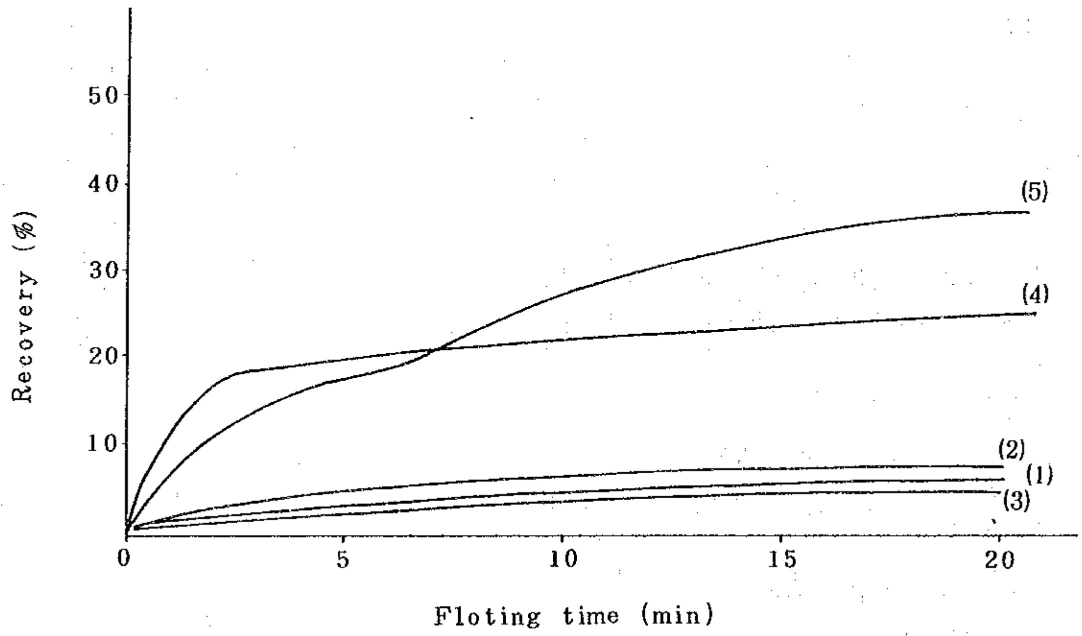
4.4.1 Comparative test for collectors

As a sufficiently favourable result was not obtained with a heavy-media separation or gravity concentration, a preliminary flotation test was carried out to obtain a coarser phosphate concentrate.

In order to select the most suitable collector for Chilembwe ore, the collecting properties of the following five collectors were compared.

1)	Nikko #204	:	oleic acid, Nohon-Koryo-Yakuhin Co. (Japan)
2)	Aero-Promoter #845	:	petroleum sulfonate, American Cyanamid Co. (USA)
3)	Tall Oil	:	fatty acid, Mitsui-Cyanamid Ltd. (Japan)
4)	Lilaflot OS#100	:	anionic phosphate collector, Keno-Gard. (Sweden)
5)	Lilaflot BS#130	:	ditto

Diagram: The Results of Comparative Test for Collectors



4.4.2 Coarse particle flotation

Aiming at the recovery of a coarser phosphate concentrate, the following flotation schemes were applied to a −14/+35 mesh fraction of feed samples conditioned under a high pulp density (40 ~ 70% solid).

Although this scheme did not bring good results owing to the instability of the froth layer, the recovery rates were low.

The range of the feed size was extended to a −14/+150 mesh, but the recovery rates were low.

The feed size range was further extended to all of the −14 mesh. As a result, the phosphorous recovery was somewhat improved. But some coarse apatite particles are still recognized in the tailings. Certainly the existence of fine particles in the flotation feed seems to help the coarse particles float by stabilizing the pulp.

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4.4.3 Semi-coarse particle flotation

The feed sample was sieved with a 28 mesh screen, then a +28 mesh fraction was ground by a rod mill. The minus 28 mesh fraction and the mill product were combined to make a flotation feed sample.

Both the concentrate grade and recovery rate are improved significantly compared with the preceding tests. But trouble is foreseeable in regard to stickiness of the froth.

4.4.4 Overall flotation test

From the results obtained by the preliminary tests, the flowsheet of the overall flotation was determined, for a case of screening with a 150 mesh screen.

Scheme I : screening with 28 mesh
Scheme II : screening and with regrinding

The results are shown in Table 4-2.

The regrinding of rougher concentrate has little effect on the rate of recovery of the final concentrate. However, the regrinding has some positive effect on a final concentrate grade.

With regrinding, all the concentrates have a grade higher than 30% P_2O_5, generally exceeding 35%. On the contrary, this process has an unfavourable effect on the production of the -150 mesh fraction. On the average, the regrinding increases the -150 mesh fraction by about 4%. Although the regrinding of rougher concentrate has some bad effect on the production of fine particles, this process may be indispensable to maintain the concentrate grade.

4.4.5 Medium size flotation test

A feed sample for this test was prepared by combining a sample from No. 2 Orebody with one from No. 4 Orebody. The combined size was -14 mesh, its weight 27 kg. The sample was then fed to a 300 × 300 mm F.W. flotation machine (one-cell). The feed sample was sieved with a 48 mesh screen, and the +48 mesh fraction was ground by a rod mill.

The minus 48 mesh fraction and mill product were fed to the flotation machine.

The results are given in Table 4-3.

The grade and recovery of the rougher concentrates showed fairly good results; it grades 38.62% P_2O_5 with an 88.0% recovery.

The final concentrates including the concentrate from slime flotation and scavenger flotation have a grade of 33.5% P_2O_5 with 99.1% recovery.

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Table 4-2. The Result of Overall Floatation Test

Test No.		P_2O_5 %				Recovery %		
		Feed	Rougher c.	Cleaner c.	Tailing	WT	R.C.	Cl.C.
Without regrinding	Z-1	21.31	33.51	34.56	8.49	51.3	80.6	80.0
	Z-2	18.30	27.14	29.56	4.56	60.8	90.2	89.7
	Z-3	12.87	28.08	30.16	1.89	41.9	91.5	91.1
	Z-4	20.47	32.29	34.36	5.54	55.8	88.0	86.8
	Z-5	20.11	33.40	34.72	5.53	52.3	86.9	85.2
	Z-6	16.85	29.71	31.77	4.07	49.9	87.9	84.8
	Z-7	18.23	33.30	35.01	10.12	35.0	63.9	63.4
	Z-8	16.53	26.82	29.59	3.35	56.2	91.9	89.2
With regrinding	Z-9	21.19	33.41	36.81	8.58	50.8	80.1	78.1
	Z-10	18.48	28.28	36.96	4.36	59.0	90.3	87.2
	Z-11	12.24	24.98	31.20	1.16	45.5	92.8	92.0
	Z-12	20.59	33.65	38.81	7.26	50.5	82.6	79.6
	Z-13	19.73	30.73	35.95	5.38	56.6	89.4	86.3
	Z-14	17.18	30.14	34.42	3.73	50.9	86.1	87.5
	Z-15	19.57	33.24	37.26	5.53	50.7	86.1	85.0
	Z-16	17.89	30.11	37.13	3.54	54.0	90.0	88.0
Ref.	F201	21.56	29.82	–	5.31	66.3	91.7	–
	F202	21.56	30.30	–	6.09	63.9	89.8	–

Remarks: Z-1&9 (No. 2), 2&10 (No. 4), 3&11 (Weathered), 4&12 (No. 2 3:1 No. 4), 5&13 (No. 2 + 10% W), 6&14 (No. 4 + 10% W), 7&15 (No. 2 + 20% W), 8&16 (No. 4 + 20% W), F201&202 (8 Min. & 5 Min. Grind)

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Table 4-3 The Result of Medium Flotation Test

Circuit	Product	Weight (g)	% P_2O_5	P_2O_5 (g)	Recovery % Wt.	Recovery % P_2O_5
Rougher (1)	Feed	23,877	17.55	4,191	88.4	89.9
	Conc.	10,630	38.62	4,105	39.4	88.0
	Tail.	13,247	0.65	86	49.0	1.9
Slime flot. (2)	Feed	3,123	15.15	473	11.6	10.1
	Conc.	1,577	29.17	460	5.9	9.8
	Tail.	1,546	0.86	13	5.7	0.3
Scavenging (3)	Feed	13,247	0.65	86	49.0	1.9
	Conc.	1,590	3.68	58	5.9	1.3
	Tail.	11,657	0.24	28	43.1	0.6
Total (1+2+3)	Feed	27,000	17.28	4,664	100.0	100.0
	Conc.	13,797	33.51	4,623	51.2	99.1
	Tail.	13,203	0.32	42	48.8	0.9

4.5 Miscellaneous Tests

Some tests were carried out to obtain data for the design of the concentrator.

4.5.1 Work index

The work index of the feed sample was measured by the Japanese Industrial Standard, to obtain designing data for grinding and crushing. The results are as follows:

Sample	Work index (kWh/T)
No. 2 Orebody	12.0
No. 4 Orebody	9.1

4.5.2 Thickening Test

The settling velocity was measured to design the thickener. The results are as follows:

Sample*	Settling velocity	Pulp density
Apatite conc.**	1.38 m/hr	49.6%
Rougher tail.	0.055 m/hr	43.3
Slime	0.357 m/hr	6.9

* These samples were obtained from medium sized laboratory tests.

** 15 mg/ℓ cations flocculant was added.

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Using sodium silicate as the dispersant and depressor for quartz, the settling rate of the rougher tailings was so small that the installment of the thickener, prior to scavenging, would be costly.

4.5.3 Filtration test

Preliminary test results suggested the need of a flocculant, for sedimentation of concentrate solids on the filter's leaf.

In order to select a suitable flocculant, three kinds of reagents were compared.

Consequently, the cationic reagent was the most effective, and its dosage was 15 mg/ℓ.

The leaf tester has a 60 mmφ "Vinylon" filter. The results of filtration test are as follows:

Pulp density	64.9%
Filtering	60 seconds
Vacuuming	30 seconds
Thickness of cake	1.83 ~ 2.10 cm
Moisture of cake	14.2 ~ 15.3%

4.6 Expected Performance

Based on the results of the overall flotation test, the medium sized laboratory test and the estimated feed grade from the calculation of minable ore, the following performance can be expected for Chilembwe ore.

Kind of ore	Assay %	Recovery %	
	P_2O_5	Wt.	P_2O_5
Feed	11.50	100.0	100.0
Conc.	30.07	33.8	88.4
Waste	2.01	66.2	11.6

Comparing this performance with that of the "Interim Report", the increase in recovery was about 5%.

The increase is made by the introduction of slime flotation. In the other report, P_2O_5 in the slime was disposed into the waste.

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5. MINING

The production scale is base on;

(1) the estimation that the required amount of P_2O_5 is 20,000 T/year in Zambia.

(2) the proposed production which amounts to 50% of the annual required P_2O_5.

(3) the limited capacity for the water storage of Mankwala Dam at the end of the dry season.

Calculation:

Minable ore grade	11.5%
Processing recovery	88.4%
Required production	$(20{,}000 \times 0.5) \div (0.884 \times 0.115)$ = 98,400 T/year (378.5 T/day)
Proposed production	400 T/day $\times$ 260 days/year = 104,000 T/year

It is impossible to obtain a scale merit because of the limited dam capacity. (The storage capacity of Mankwala Dam at the end of the dry season; 225,000 T) However, recycled water is used as much as possible. (see Chap. 6 & 7)

5.1 Mining Method

Open pit mining is selected for the following reasons:

(1) The deposits are found near the surface.

(2) The maximum depth of the deposits is about 60 m from the surface.

(3) An adequate waste dump area is available nearby.

(4) The stripping ratio is within an economical allowance.

(5) Rain falls for only a short time even during the rainy season.

5.2 Selection of Equipment

Selection of equipment as to the make, model and size, and determination of the number of required units is based on their performances. Small size equipment is selected to suit the pit size and the scale of production.

5.3 Ultimate Pit Design (Refer to Fig. 5.1 ~ 5.4)

The following criteria are used in designing the ultimate pit.

(1) The final pit slope of 45° is decided because the geotechnical data is limited. However, the study of slope stability should be continued during the mining practices to seek an applicable steeper angle for the pit slope to minimize the overall stripping ratio and the operating costs.

(2) A bench height of 5 m and its slope of 70° are selected, considering the slope stability, the waste-ore ratio and the performance of the 2.2 cu.m. dozer shovel.

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(3) Ore reserves of No. 2 Orebody and the shallow part of No. 4 Orebody are chosen for the present minable ore calculation. These figures are then converted to those of minable ore reserves, using the criteria as mentioned above: 1,551,000 T with an average grade of 11.5% P_2O_5 and a stripping ratio of 2.16 (Refer to Table 5).

5.4 Mining Plan

The mining term is divided into the following three stages:

(1)	Preparatory Stage	Pre-production stripping
(2)	Production Stage I	Production from the No. 2 Orebody above the 860 m level.
(3)	Production Stage II	Production from the No. 2 & No. 4 Orebody.

(1) Preparatory Stage

To avoid the rainy season, pre-production stripping will start from the first year prior to the commencement of production.

Within a year, an area for extracting crude ore for a 6 month period can be exposed. The amount of pre-production stripping is determined for minimizing the initial investment. The total volume of overburden to be removed is 124,000 cu.m., containing 52,000 T of ore which will be stockpiled separately for trial production.

(2) Production Stage I

This stage will last for 5 years. An average grade during this period is estimated to be 11.5% P_2O_5 with a stripping ratio of 2.1.

(3) Production Stage II

This stage will last for 10 years. An average grade during this period is estimated to be 11.5% P_2O_5 with an overall stripping ratio of 1.95 (2.1 from 8th to 15th year, 1.83 in 16th and 0.27 in 17th year).

Exploration data indicates the existence of other orebodies such as No. 1 and No. 3 Orebodies and the deep horizons of No. 4 Orebody.

Due to the limitation of available data, these deposits are excluded from the present evaluation. However, the No. 1 Orebody seems to be minable without difficulties to a certain depth within an allowable waste-to-ore ratio.

The No. 3 Orebody has a narrow and steeply dipping dyke form. As this orebody is of a high grade of ore, it may be possible to extract only a shallow part of this deposit, which helps easier control of the grade of feed even with its lesser amount. In this case, an appropriate pit slope should be determined through the mining practice, although an applicable steepness will limit the minable ore amount that can be economically extracted.

5.5 Operational Plan

For the concrete supply of ore and maintaining the grade to the concentrator, it is preferable to increase the number of benches. The greater the number of benches, the easier it is to control the ore grade. However, increasing the number of benches increases the initial investment and operating costs by rising the waste ore ratio. Therefore, a three bench operation system is selected.

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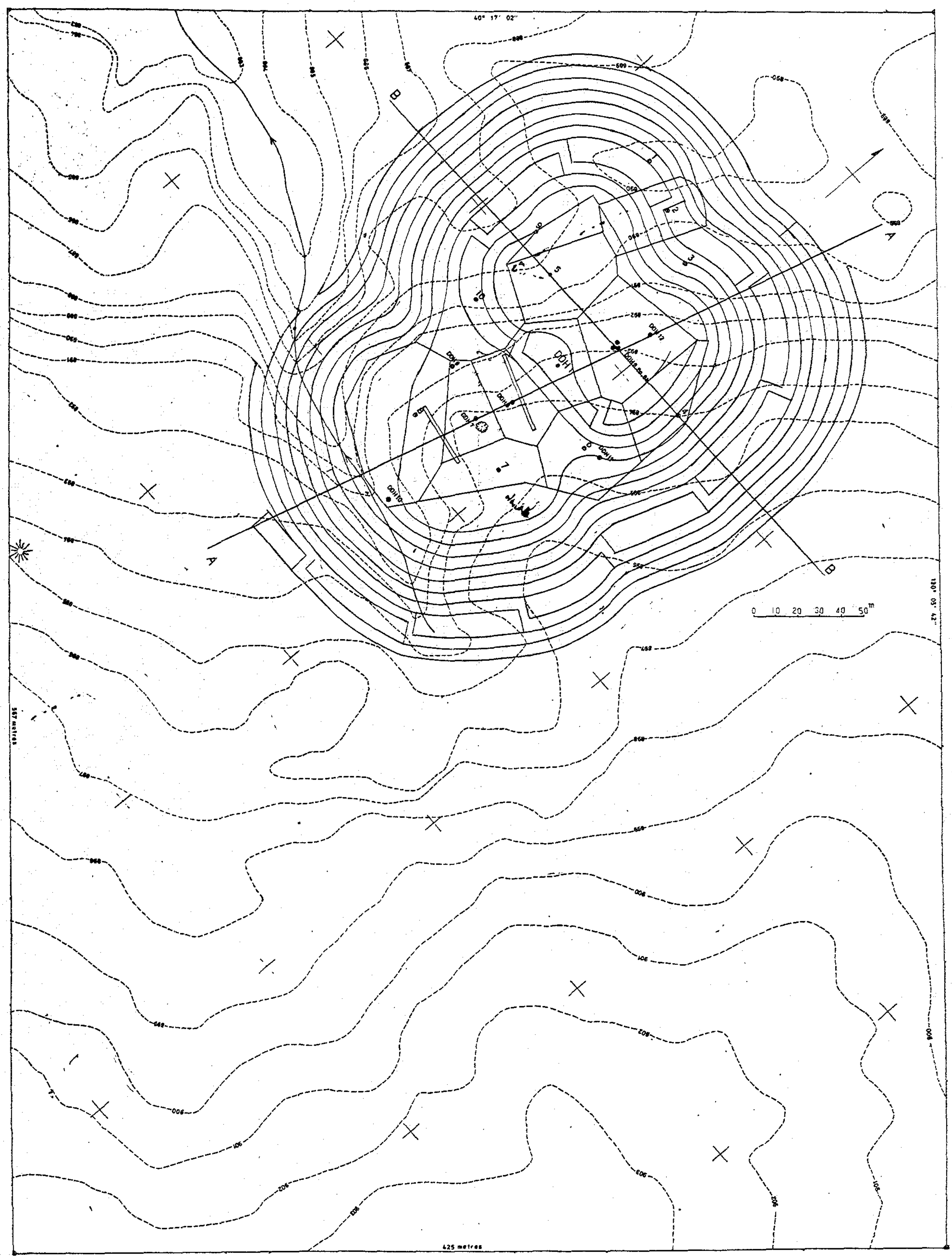
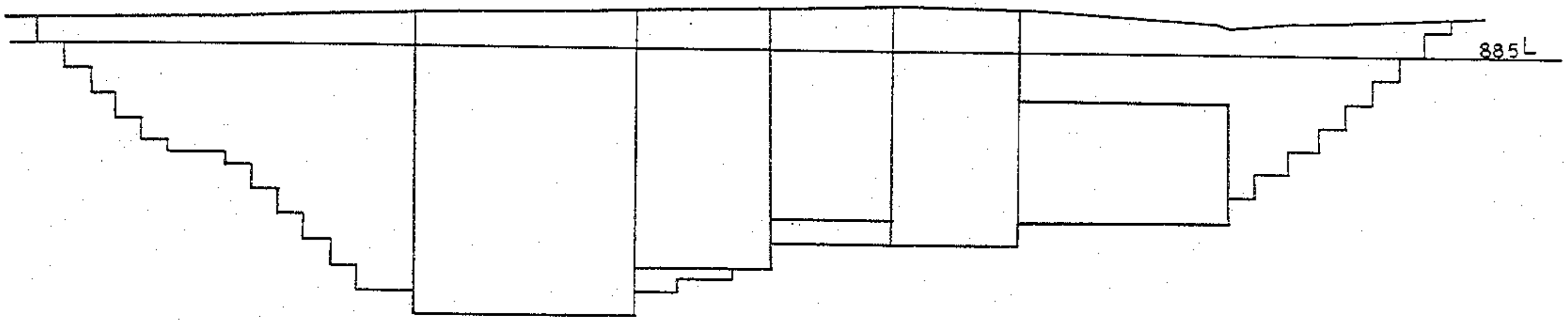


Fig. 5.1 Final Pit Design for No.2 Orebody. (1)

A—A SECTION



B—B SECTION

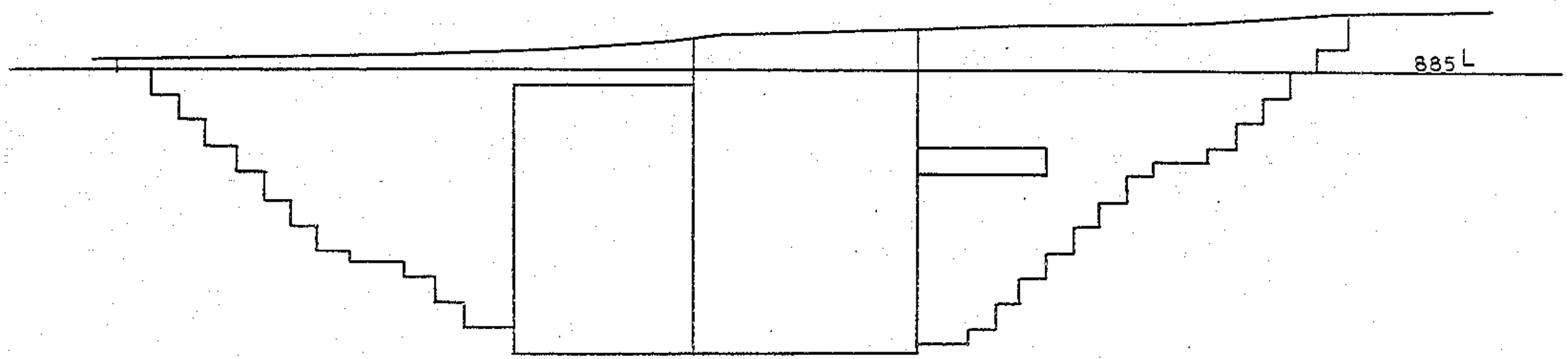


Fig. 5.2 Final Pit Design for No.2 Orebody (2)

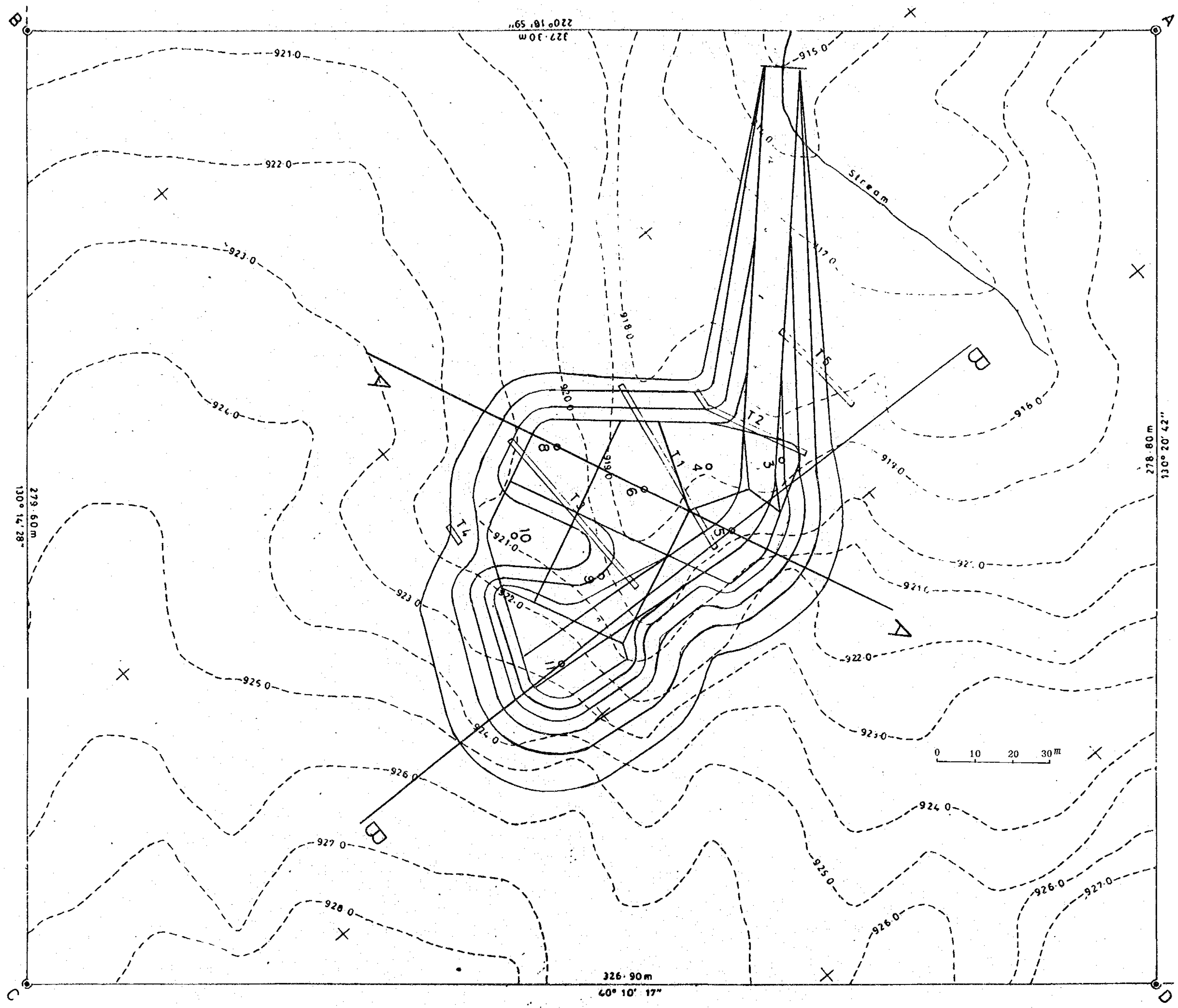


Fig. 5.3 Final Pit Design for No.4 Shallow Orebody. (1)

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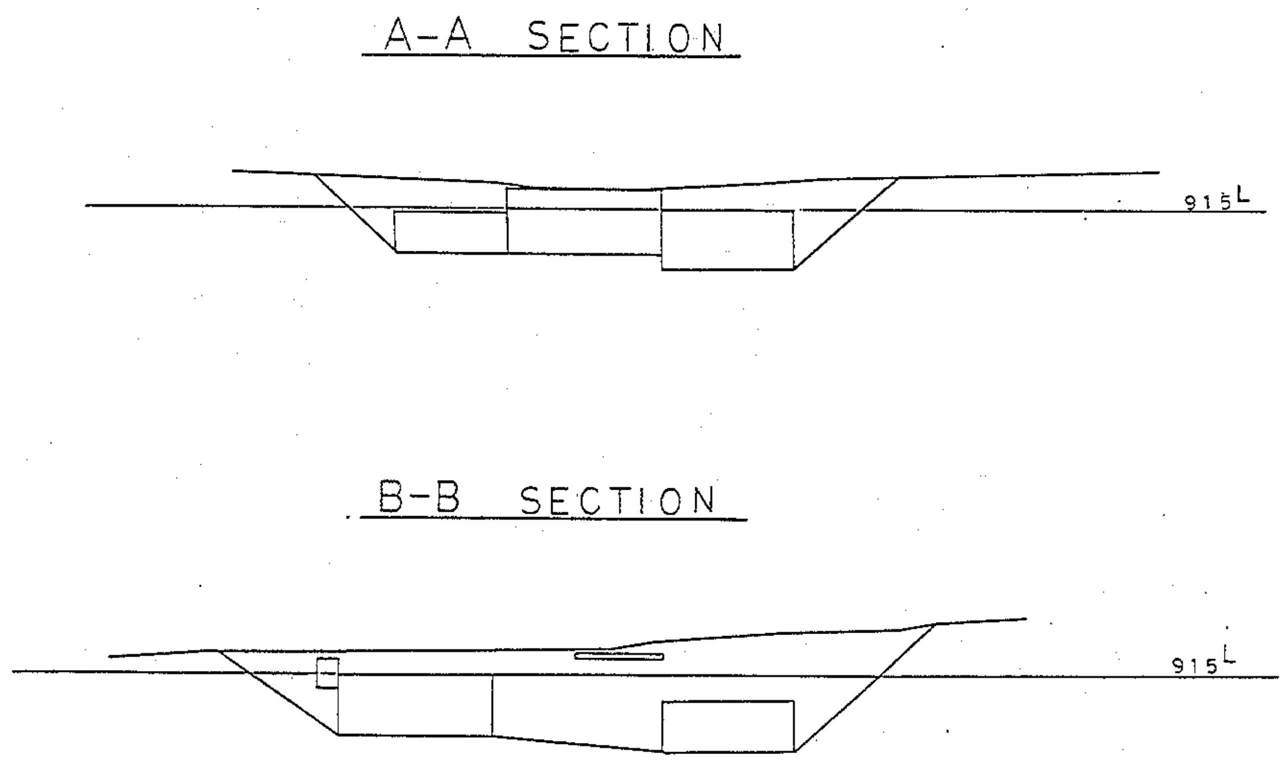

Fig. 5.4 Final Pit Design for No.4 Shallow Orebody (2)

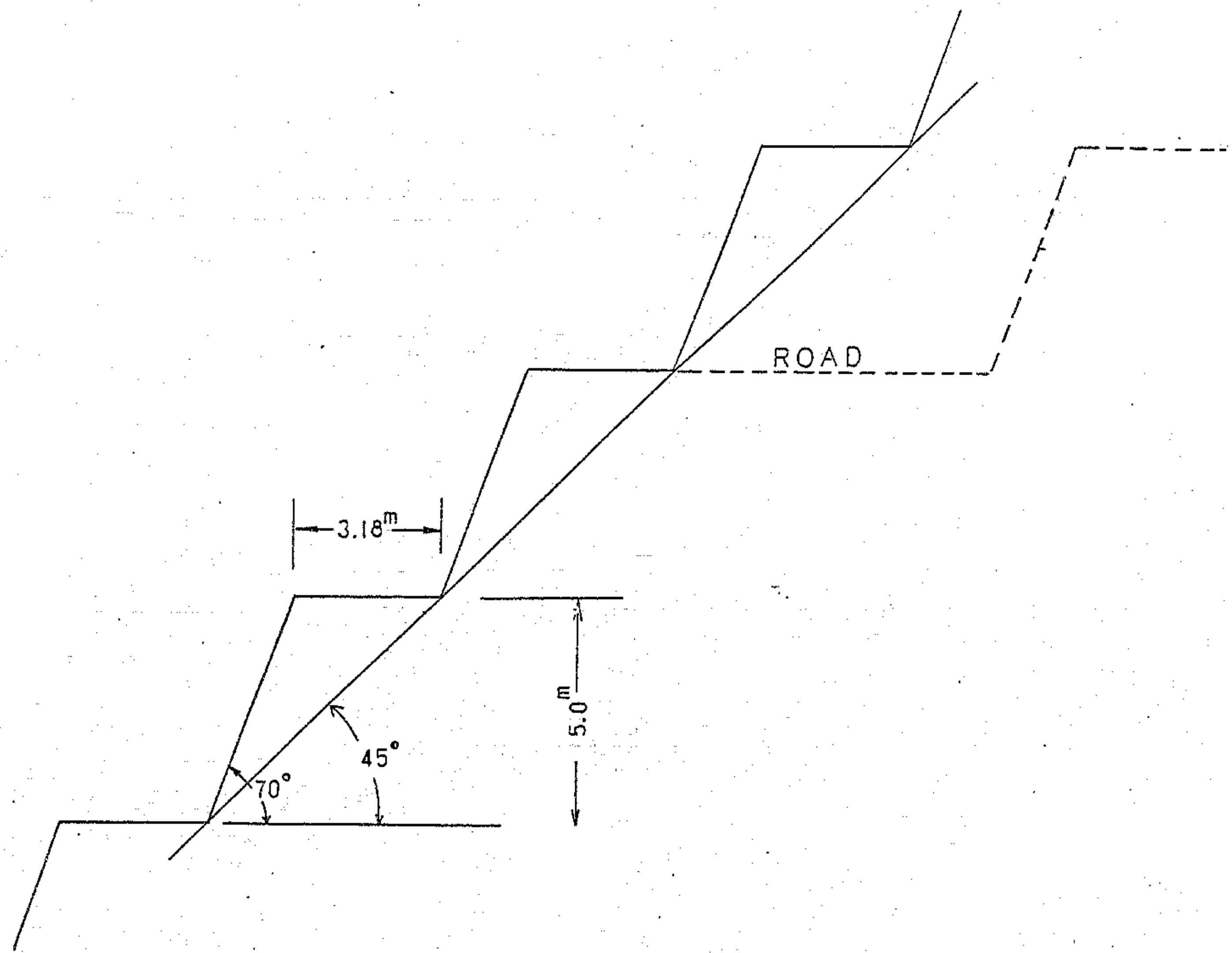

Fig. 5.5 Final Bench Slope

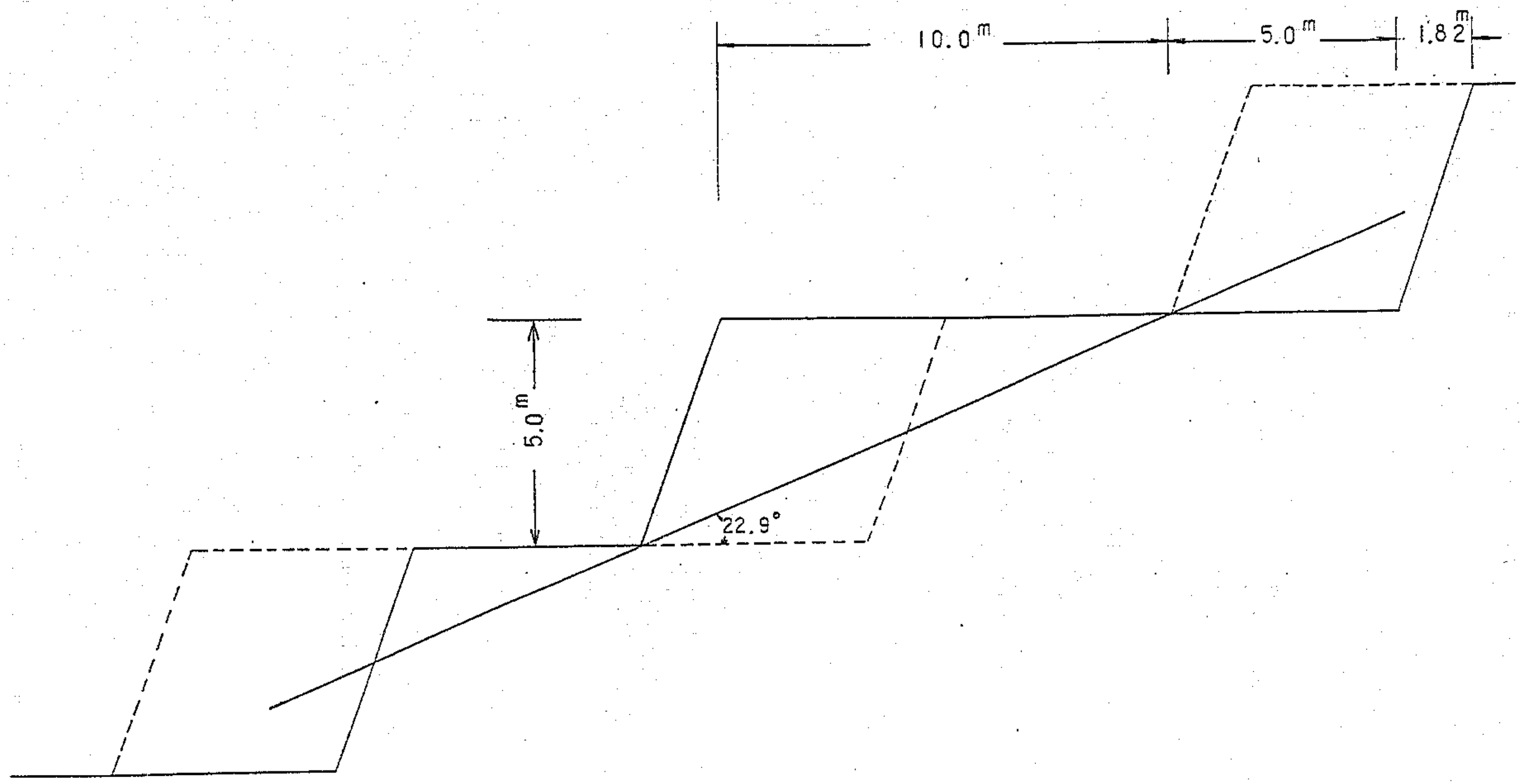

Fig. 5.6 Working Bench Slope

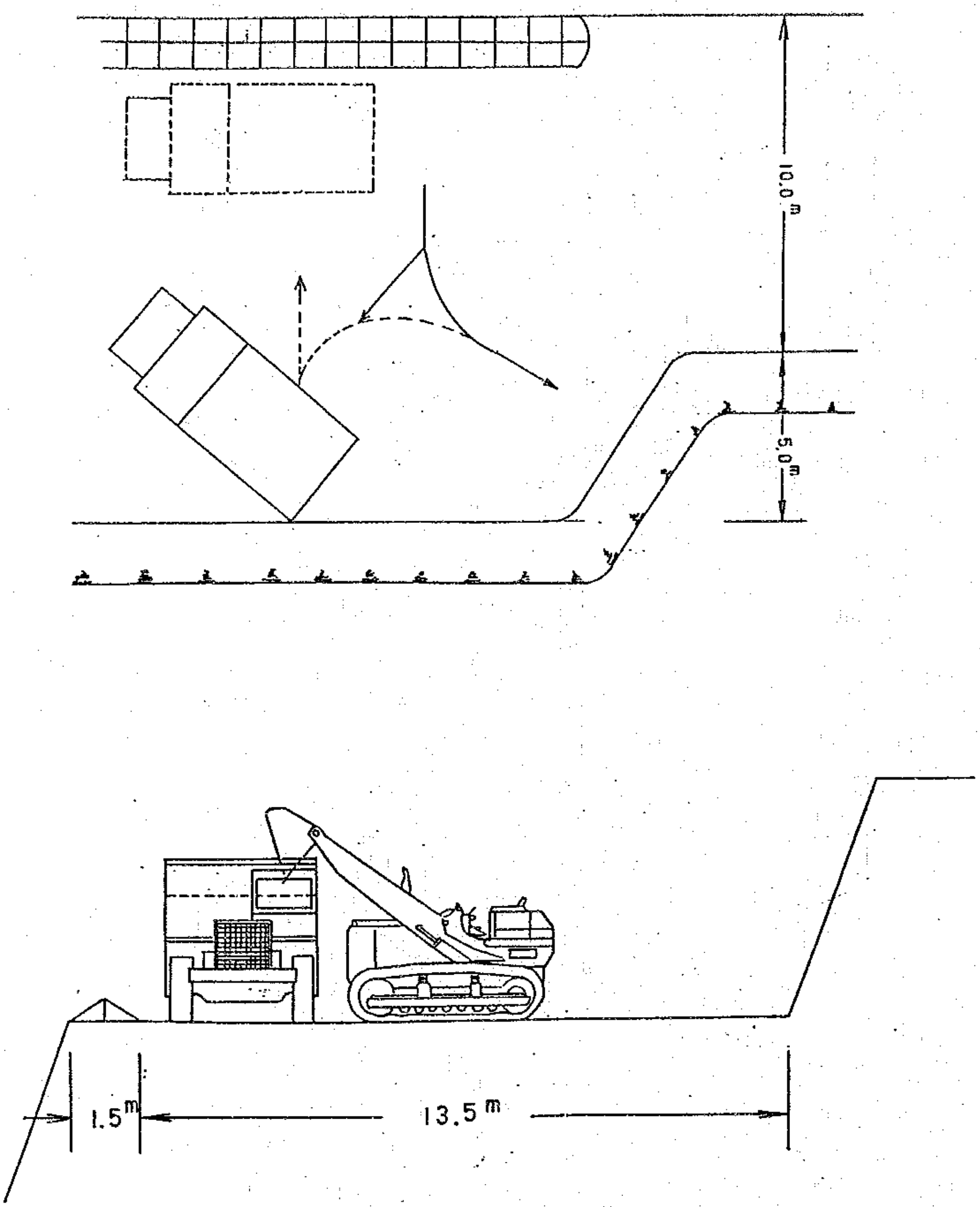

Fig. 5.7 Relationship of Equipment Size, Benchspacing and Operating Room for 20 ton Truck and 2.2 cu.m Dozer Shovel

Table 5.1. Minable Ore Reserve (No. 2 and No. 4 shallow Deposits)

( ) ore grade %

Block \ Level	Surface ~885L	~880	~875	~870	~865	~860	~855	~850	~845	~840	~835	~830	Total
DDH 6	(17.8) 23,410	(12.9) 13,770	(7.3) 13,770	(9.4) 13,770	(15.2) 13,770	(13.2) 13,770	(13.2) 13,770	(13.2) 13,770					(13.2) 119,800
DDH 7	(9.7) 22,630	(9.4) 12,570	(2.7) 12,570	(17.1) 12,570	(11.4) 12,570	(11.4) 12,570	(8.6) 12,570	(6.7) 12,570	(5.3) 12,570				(9.2) 123,190
DDH 9	(13.0) 41,180	(11.1) 25,740	(16.2) 25,740	(19.3) 25,740	(13.3) 25,740	(12.4) 25,740	(20.0) 25,740	(14.8) 25,740	(11.8) 25,740	(7.7) 25,740	(9.5) 25,740	(10.0) 25,740	(13.2) 324,320
DDH 11		(11.9) 15,330	(8.6) 15,480	(8.5) 15,480	(5.3) 15,480	(11.9) 15,480	(8.0) 15,480	(11.1) 15,480	(9.3) 15,480	(9.3) 6,190			(9.3) 129,880
DDH 12				(8.0) 5,700	(8.0) 5,700	(8.0) 6,040							(8.0) 17,440
II 2		(16.2) 8,400	(9.5) 16,470	(8.0) 16,470	(15.6) 16,470					(5.3) 5,280			(11.2) 63,090
II 4					(7.0) 14,880								(7.0) 14,880
II 5		(20.0) 7,940	(9.9) 19,370	(14.0) 19,370	(7.8) 19,370	(2.9) 19,370	(5.0) 19,370	(29.1) 19,370	(27.1) 19,370	(24.4) 19,370	(21.3) 19,370	(25.4) 21,300	(16.9) 203,570
II 6		(13.2) 15,780	(7.2) 23,420	(12.6) 23,420	(11.9) 23,420	(8.8) 23,420							(10.6) 109,460
II 7	(16.8) 9,800	(7.3) 13,880	(8.8) 13,880	(15.3) 13,880	(14.2) 13,880	(9.4) 13,880	(13.5) 13,880	(10.0) 13,880	(16.6) 13,880	(12.9) 13,880			(12.3) 134,720
II 8				(9.6) 38,130	(4.8) 38,130	(5.1) 38,130	(10.6) 38,130	(6.4) 38,130					(7.3) 190,650
II 10							(5.7) 11,720						(5.7) 11,720
IV (No. 4 Orebody)	(10.0) 108,260												(10.0) 108,260
Total	(11.8) 205,280	(12.1) 113,410	(9.4) 140,700	(12.4) 184,530	(10.0) 199,410	(7.9) 168,400	(11.2) 150,660	(12.7) 138,940	(14.6) 87,040	(13.3) 70,460	(14.6) 45,110	(17.0) 47,040	(11.5) 1,550,980T

Daily production will be 400 T; ore blasting is every three days with the other days being for waste removal.

Stockpiled ore in the yard will be used in the concentrator during the non-extractable term of the rainy season.

Drilling Blast holes of 4"ϕ will be drilled by a hydraulic crawler-drill with spacings of 2.33 and 3.0 m in two rows and to a depth of 5.71 m, including 0.71 m of sub-drilling.

Blasting Most holes are expected to be dry, and a mixture of AN-FO is used. Slurry or dynamite explosives are provided only for the blast holes that cannot be dewatered. The powder factor for 4"ϕ holes is 267 g/T, including a 10% extra for secondary blasting.

Loading Blasted ore and waste are loaded by a dozer-shovel of a 2.2 cu.m. capacity.

Hauling The 20 t rear dump trucks are the main units for hauling the ore to the primary crushing plant and the waste to the dump area.

Maintenance of all haulage roads in good condition at all times is one of the most important factors to reduce the costs of tires and truck maintenance.

A fleet of the road maintenance equipments consisting of a grader, a bulldozer and a water cart will be provided for this purpose.

Drainage The greater part of the precipitation falls in a three month season. During the pre-stripping period, the surface water is drained through ditches. For the purpose of de-watering in the pit, a pump is used below the 885 meter level.

Grade-Control The cuttings of drill holes should be collected and assayed prior to blasting, not only to discriminate ore from waste but also to find the grade of ores at each working face. This data will serve for the grade control of the daily operation.

Service Vehicle A service truck and pick-ups are provided for surveying, maintenance and supervision, etc., to support smooth and efficient operations.

Facilities Magazines to store a 6 month supply of explosives, AN-FO and detonators will be installed.

A repair shop for the heavy duty equipment will be constructed at the mine site.

5.6 Technical Data

5.6.1 Design Parameter

Mining method	Open pit
Tonnage mined	104,000 T per year (400 T/day)
Operation condition	
Working days per year	260 days
(Total days 365 less 52 Sundays, 11 public holidays, 12 Christmas holidays, 30 days unworkable in rainy season)	
Shift per day	1 shift
Hours per shift	8 hours

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Physical properties of ore and waste	
Specific gravity, Ore	3.0 T per cu.m.
Waste	2.8
Compressed strength, Ore	800 to 1,000 kg per sq.cm
Waste	800 to 1,000
Pit slope (refer to Fig. 5.5 to 5.7)	
Final pit slope	45°
Working slope	22.9° (3 bench group)
Bench	
Height	5.0 m
Width: working bench	15.0 m
final bench	3.18 m
Working bench slope	70°
In-pit road grade	max. 10%

5.6.2 Production

Total minable ore	1,551,000 T 11.5% P_2O_5
Overall waste : ore ratio	2.16 : 1
Life at 400 T per day	14.9 years

	Year 2 ~ 7	Year 8 ~ 17	Total
Mined ore	546,000 T	1,005,000 T	1,551,000 T
Average grade	11.5%	11.5%	11.5%
Waste : ore ratio	2.1 : 1	1.95 : 1	2.0 : 1
Waste + ore per day	1,240 T	1,180 T	1,200 T

Note: The waste ore ratio does not cover pre-stripping.

5.6.3 Pre-production stripping

Tonnage of surface soil	57,540 T
Tonnage of waste rock	293,130 T
Total tonnage of overburden	350,670 T
Period	1 year
Ore contained	52,000 T (equivalent to six month supply)

5.6.4 Operation plan

Primary blasting

Drill	Hydraulic crawler drill
Hole diameter	4 inch

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Hole inclination	90°
Sub-drilling	0.71 m
Total length of a hole	5.71 m
Drilling speed	17.8 m per hour
Drilling pattern	2.35 × 3.0 m
Explosives	mainly AN-FO (80%)
	slurry or dynamite for the wet holes (20%)
Power factor	0.8 kg per cu.m (267 g per T)
	including secondary blasting
Loading	
Loader	2.2 cu.m dozer shovel
Type of loading	V-shape loading
Loading capacity	294 T/hr
Hauling	
Ore and waste	20 T rear dump truck
Hauling capacity	50.56 T/hr
Drainage	
Average annual rainfall	approx. 1,000 mm
Pumping start level	885 mL
Pumping start year	year 3

5.6.5 Waste dump

Total tonnage of waste	approx. 3,346,600 T
Bulk density of dumped waste	1.56
Required dump capacity	approx. 2,145,000 cu.m

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5.6.6 Required equipment in the pit

Item	Nos.	Spec.
Drill	2	Hydraulic crawler drill Dimension 7,000 L × 2,430 W × 3,740 H
Dozer shovel	2	Bucket capacity 2.2 m^3 200 HP, 6,200 L × 2,500 W
Bulldozer	1	Blade width 4,365 mm Weight 18,850 kg
Back hoe	1	35 HP Weight 4,320 kg
Pickup	2	4 WD 3,264 cc
2 T truck	1	
Dump truck	6	Loading weight 20 T 7,450 L × 3,200 W × 3,450 H
Drainage pump	1	2.2 to 7.3 m^3/min, 40 to 80 mH
Total	17	

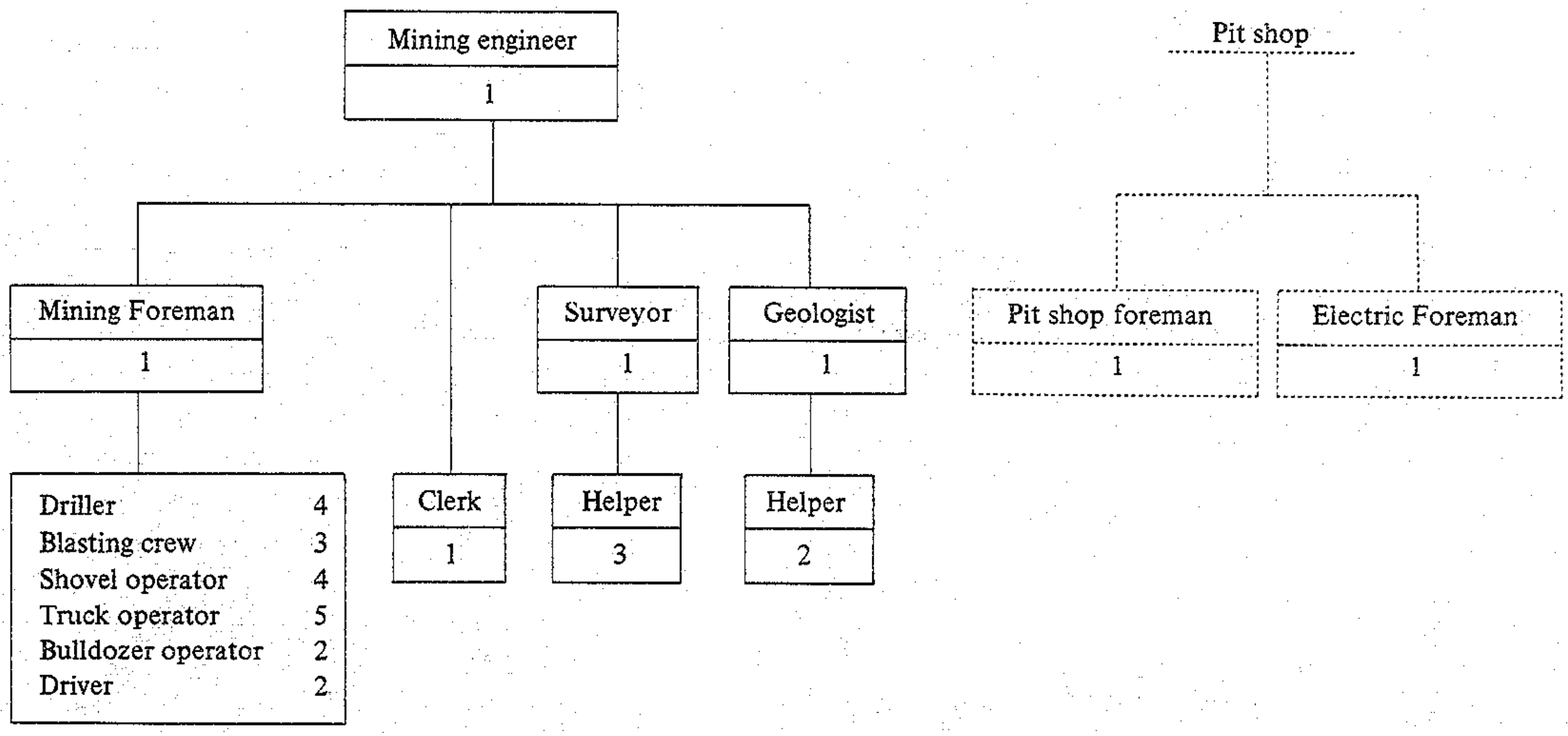
√4

5.7 Operating Cost

5.7.1 Operating Cost Total

	Item	Unit	Unit Price	Drilling		Blasting		Loading		Transportation		Face cleaning, Waste pile		Miscellaneous		Total	
			US$		US$												
Material	Bit (102 mmφ)	piece	200	88	17,600											88	17,600
	Bit (50 mmφ)	〃	125	10	1,250											10	1,250
	Rod (38 mmφ)	〃	150	35	5,250											35	5,250
	Sleeve (38 mmφ)	〃	43.2	18	778											18	778
	Dynamite	T	500			17,216	8,608									17,216	8,608
	AN-FO	T	460			68,865	31,678									68,865	31,678
	Detonator	piece	0.636			3,224	2,051									3,224	2,051
	Diesel oil	ℓ	0.56	14,685	8,224			29,290	16,402	119,250	66,780	30,903	17,306	18,044	10,105	212,172	118,817
	Lubricant				822				1,640		6,678		1,731		1,010		11,881
	Tire (16-25-24)	piece	1,172							26	30,472					26	30,472
	(9-20-12)	〃	83											11	913	11	913
	(7-16-10)	〃	76											8	608	8	608
	Stationary	lot													1,000		1,000
	Others	〃			1,696		2,117		902		5,197		952		682		11,546
	Maintenance, Repair	〃			7,780				9,576		42,557		8,764		3,992		72,669
	Total (US$/T)				43,400 (0.42)		44,454 (0.43)		28,520 (0.27)		151,684 (1.46)		28,753 (0.28)		18,310 (0.17)		315,121 (3.03)
Labour	Total (US$/T)			4	9,600 (0.09)	3	7,200 (0.07)	4	9,600 (0.09)	5	12,000 (0.11)	2	4,800 (0.05)	12	32,100 (0.31)	30	75,300 (0.72)
Total	(US$/T)				53,000 (0.51)		51,654 (0.05)		38,120 (0.36)		163,684 (1.57)		33,553 (0.33)		50,410 (0.48)		390,421 (3.75)

5.7.2 Organization



5.7.3 Material consumption

Bit	200 m/piece
	annual drilling length 17,427 m
Rod	1,000 m/piece
Dynamite	53.4 g/T
AN-FO	213.6 g/T
Detonator	0.01 pc./T
Fuel oil	
Drill	15.0 ℓ/hr
Shovel	26.7
Dump truck	18.7
Bulldozer	42.5

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6. CONCENTRATION

6.1 Outline

The concentrator is designed for all slime flotation and will treat 104,000 T in 290 working days annually producing phosphorous concentrates.

The plant is divided into three processes; the primary crushing, the secondary-tertiary crushing and the grinding-flotation-filtration.

The ancillaries like the concentrate thickner are also a part of the concentrator.

To allow time for repairs, the maximum treatment capacity of the concentrator will be 400 T/day.

The annual working days of the extraction are fixed at 260 days, because of provision against unworkable days during the rainy season. The production will be stabilized by the supply from the stock pile (26,000 T) installed between the pit and the primary crushing plant.

The location of the concentrator is decided, considering the topography, the foundation of the rock, the direction of the wind, the ultimate pit location and the tailing pond location.

6.2 Primary Crushing Plant

The run of mine will be transported from the pit by a 20 T truck to the primary crushing plant where it will pass over a grizzly with 100 mm spacing.

The oversize ore is crushed by a primary single toggle crusher and combined with the undersize ore from the grizzly. Then the ore will be transported by a belt-conveyor to the coarse ore storage bin.

6.3 Secondary-Tertiary Crushing Plant

The ore from the coarse ore storage bin is drawn by an apron feeder and transported to a single deck secondary vibrating screen with 60 mm openings by a belt-conveyor.

The oversize ore is crushed by a secondary single toggle crusher and transported to a single deck tertiary screen with 15 mm openings. A single toggle crusher will be selected to reduce the initial costs.

For washing, the undersize ore of the secondary screen is fed to a double deck vibrating screen with 15 mm and 5 mm openings by a belt-conveyor.

The +15 mm ore is transported to a tertiary screen by a belt-conveyor, while the −15/+5 mm ore is transported to the 400 T fine ore storage by a belt-conveyor.

The −5 mm ore (pulp) is fed to a spiral classifier.

The rake up sand from the classifier is transported to the fine ore storage by a belt-conveyor, and the overflow from the classifier is sent to the flotation plant by a pump.

The oversize ore from the tertiary screen is crushed by a cone crusher and this product is recycled to the tertiary screen by a belt-conveyor. The undersize ore from the tertiary screen is transported to the fine ore storage.

6.4 Grinding

The grinding circuit comprises a ball mill and a spiral classifier which is operated in a closed circuit of a wet grinding system. From the fine ore storage, the ore is drawn by a belt feeder and fed to the ball mill by a belt-conveyor. The 80% pass size of the ground ore is 0.30 mm and the pulp density is 50% solids.

6.5 Flotation

The flotation circuit is comprised of a rougher, a scavenger, a cleaner and a slime flotator. The flotator cells are types #21 and #24 Fahren Wald flotation machines.

The tailing of the rougher is fed to the cyclone, the under flow is fed to the scavenger and the overflow becomes the tailings, because it is degraded by the rougher flotation.

The required amount of the flotation reagents is low in high pulp density, therefore it means a lower operational cost.

The froth of scavenger is sent back to the rougher and the trailing becomes the final tailing.

From the crushing circuit, the slime is fed to a cyclone. For the recovery improvement the under flow is fed to the slime flotation, and the clayey overflow is eliminated, because it causes some bad influences on the flotation.

6.6 Thickening and Filtering

The P_2O_5 concentrate froth is thickened in a 10 mϕ thickener, then sent to the filtering plant.

The thickened concentrate is filtered by a drum-filter and the filtered cake is stored in the 500 T stock yard for concentrates.

The concentrate is loaded by a shovel loader and transported by trucks after being weighed by a truck scale.

6.7 Tailing

The tailing of the scavenger is sent to a tailing pond by a pump, and the thickener overflow, the tailing of slime flotation and the cyclone overflow are sent to the pond by another pump.

The overflow of the tailing pond is re-cycled and re-used in the concentrator.

6.8 Tonnage and Grade Control

The tonnage of the run of mine is weighed by the conveyor scale at the primary crushing plant.

The product from the concentrator is calculated by the truck scale.

The final tailing is then claculated by comparing the crude ore and the product.

The grade control is obtained by the samples of the final concentrate and the final tailing, according to which the crude ore grade is calculated.

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6.9 Process Water

The process water consists of the overflow of the tailing pond and the fresh water.

To recover the used water as much as possible because of water shortage in the area, the amount of re-used water should be increased.

The recycled water recovery from the tailing pond is possible because;

(1) The settling velocity of fine particles is so slow that the big thickner is necessary to settle the fine particles. The tailing pond serves this purpose.

(2) The reagent in the water is decreased by the absorption, resolution and dilution in the tailing pond.

(3) The tailing pond is constructed close to the concentrator.

6.10 Reagents

The reagents for the flotation process are caustic soda, Lilaflot and water glass.

The caustic soda is dissolved in the reagent room, fed to the conditioner and controlled by a pH meter.

Other reagents are dissolved and adjusted in a reagent room adjacent to the concentrator and fed to the flotation process by a remote controlled feeder.

6.11 Instrumentation

Instruments for monitoring, indicating and recording will be installed at each necessary point.

(1) For the weighing of ore drawn from storage, three weight meters will be installed.

(2) For the determination of the volume in the washing and grinding section, two flow meters will be installed.

(3) For the measurement of a pulp pH value in the flotation process, two pH meters will be used.

(4) For the determination of pulp dennity in the grinding section, one density meter will be installed.

6.12 Miscellaneous

(1) For the detention of dust in the crushing section, two dust arresters will be installed.

(2) For the sampling of final concentrates and final tailings, two samplers will be installed.

(3) A sequence control will be used for the starting up machines in all sections.

(4) Three overhead craines will be provided for the maintenance, repair, and handling of mechanical equipment in the crushing plant and the concentrator.

(5) Store rooms will be provided in each section.

(6) An office will be provided in the concentrator.

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6.13 Analysis and Laboratory

An analysis and laboratory room will be built adjacent to the concentrator, these facilities will be used for other purposes as well as for the concentrator.